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MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO

BY
TRYGVE D. YENSEN
AND
WALTER A. GATWARD

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MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO

BY
TRYGVE D. YENSEN
RESEARCH ASSISTANT PROFESSOR OF ELECTRICAL ENGINEERING
AND
WALTER A. GATWARD
RESEARCH FELLOW IN THE ENGINEERING EXPERIMENT STATION

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MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO

I. INTRODUCTION

IT is the purpose of this bulletin to record the results of experiments to determine the magnetic and allied properties of iron-aluminum alloys melted in vacuo. Previous investigations to determine these properties of pure iron,* iron-boron alloys,† and iron-silicon alloys, ‡ melted in vacuo, have been reported in earlier bulletins. Because of the similarity of many properties of aluminum and silicon, the same remarkable results were expected from the iron-aluminum as from the iron-silicon alloys. To some extent these have appeared. The results obtained, although not covering every phase of the subject, include the magnetic, electrical, and mechanical properties. Chemical analysis and photomicrographs are also presented for a number of the alloys.

The chemical analysis was in charge of Mr. J. M. Lindgren of the Chemistry Department; the methods used for the determination of aluminum are described by Mr. Lindgren in the Appendix. The photomicrographs were prepared by Mr. F. E. Rowland, also of the Chemistry Department. Mr. H. R. Fritz, Research Fellow in the Engineering Experiment Station, has rendered valuable service throughout the year as general assistant in connection with the investigation. Besides acknowledging their indebtedness to those persons who have been directly connected with the work, the authors also wish to express their appreciation to many others who have been of service to them. Among these it is particularly desired to mention Professor E. B. Paine, acting head of the Electrical Engineering Department.

Aluminum was first used in the industries in connection with copper to form aluminum bronzes, and it was this application of it that stimulated investigators to develop methods of manufacture capable of producing it at a cost which would warrant its use in large quantities. It is well known today that aluminum occurs in nature only in the form of oxides and that it is one of the stablest oxides to be found, so stable that the only way of reducing it to its metallic state is by means of electrolysis. This fact early suggested its use for the purpose of reducing other oxides, and it was in the capacity of a reducer, deoxidizer, or degasifier, that it very early became the friend of the iron and steel maker. For this purpose aluminum soon became a serious competitor of silicon, in spite of the high cost of aluminum in its early days. Many were the wonderful qualities attributed to this new metal. A large number of these, however, existed only in the imagination of their advocates and were soon disproved.

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*"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914. Trans. Am. Inst. Elect. Engrs., vol. 33, I, p. 451.

†"The Effect of Boron Upon the Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 77, 1915.

‡"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915. Proc. Am. Inst. Elect. Engrs., vol. 34, p. 2455. Oct., 1915. Proc. Am. Inst. Mining Engrs., p. 482, Feb., 1916.

Among the well-established properties of aluminum, the following are perhaps the most important: *

a. Aluminum when added to molten iron will reduce iron oxide to metallic iron. Furthermore, it will reduce CO or CO_2 gases to free carbon and thus degasify, quiet the bath, and prevent blowholes during solidification and cooling. If the iron is sufficiently liquid, the Al_2O_3 formed will pass to the surface; if it is not, the Al_2O_3 will become entangled in the iron, increase the viscosity, and cause the iron to be more or less unsound.

b. When added in sufficient quantities, 2 to 5 per cent, aluminum will cause carbon to be completely precipitated as graphite and thus transform, for example, white cast iron into gray iron. If 10 to 20 per cent aluminum is added, however, the carbon may all be in the combined form Fe_3C.

c. As the heat of combustion liberated by the oxidation of aluminum is much greater than that absorbed by the reduction of iron oxides, clearly illustrated by the use of thermite in welding iron, the addition of aluminum to oxidized iron or steel causes a rise in the temperature of the bath. This fact, and not any appreciable lowering of the melting point, is the cause of the higher fluidity noticed when a small percentage of aluminum is added to the bath.†

The metallographic investigation of iron-aluminum alloys was first systematically undertaken by the Alloys Research Committee of the Institution of Mechanical Engineers,* and later, more extensively, by Gwyer.† The equilibrium diagram according to the latter is shown in Fig. 1. This diagram shows that iron and aluminum, when liquid, dissolve in each other in any proportion. Up to an aluminum content of 50 per cent mixed crystals are at first formed upon solidification, but as the solid solution becomes saturated with

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*Birmingham Eng. Jour., vol. 3, pp. 138-145. Transactions Am. Inst. Min. Engrs., vol. 18, pp. 102-122, 1889-90. Journal Iron and Steel Inst., vol. 37, I. pp. 112-133, 1890. Transactions Am. Inst. Min. Engrs., vol. 18, pp. 835-58, 1889-90. The Eng. and Min. Jour., vol. 50, p. 213, rev. vol. 42, p. 265. Iron & Steel Trades Jour., vol. 43, pp. 309, 399. Transactions Am. Inst. Min. Engrs., vol. 20, p. 233, 1891. Jour. Soc. of Chem. Ind., vol. 12, p. 239, 1893.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

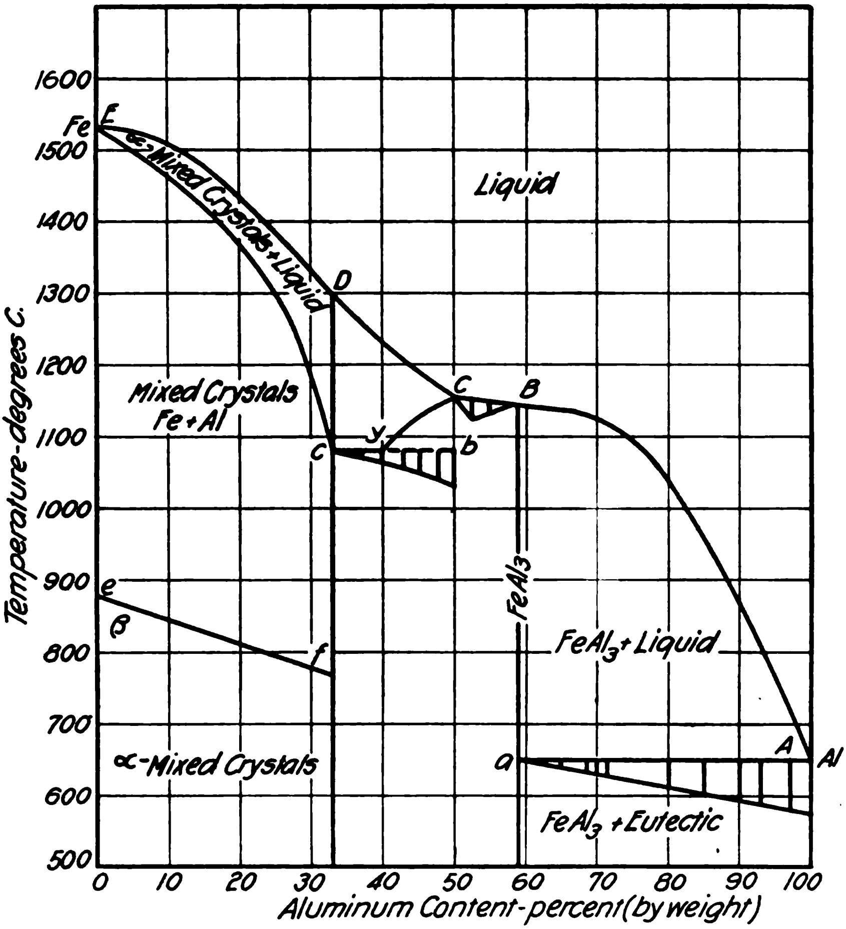

FIG. 1. EQUILIBRIUM DIAGRAM OF IRON-ALUMINUM ALLOYS ACCORDING TO GWYER

an aluminum content of 34 per cent, any excess above this amount will be precipitated as solidification proceeds. Alloys containing less than 34 per cent aluminum will suffer no precipitation of any constituent during cooling and, therefore, at ordinary temperatures will consist of mixed crystals of iron and aluminum. The lowering of the melting point of iron due to small percentages of aluminum is very slight. Even for a 10 per cent aluminum content the melting point is lowered only about 10 degrees. This shows conclusively that the increased fluidity of iron, previously referred to, which is obtained by adding less than 0.1 per cent aluminum, can not be due to the lowering of the melting point. With an aluminum content of 59 per cent, the compound $FeAl_3$ crystallizes out during solidification and leaves behind liquid aluminum, which freezes at 653 degrees. The theory suggested by previous writers, that the liberation of heat noticed upon adding a small amount of aluminum to molten iron might be due to the formation of compounds of iron and aluminum,* is, therefore, at variance with the evidence just presented. The only explanation of the described phenomenon which has stood the test of investigation is that the heat liberated is due to the formation of Al_2O_3.

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*Rpt. Alloys Research Committee, Proc. Inst. Mech. Engrs., No. 2, pp. 238-297, 1805.

†Z. Anorg. Chem., vol. 57, pp. 113-53, 1908.

From these investigations it appears that within the region studied in the present research, namely 0 to 10 per cent aluminum, the alloys all consist of mixed crystals of iron and aluminum, just as the iron-silicon alloys described in Bulletin 83 all consisted of mixed crystals of iron and silicon. The photomicrographs of Gwyer, Guillet, and Hadfield confirm the deductions just stated.

The mechanical properties of iron-aluminum alloys have been studied by Hadfield,† Styffe,‡ and Guillet,§ who all agree that aluminum in quantities up to 5 per cent has very little influence upon the mechanical properties of iron. When more than 5 per cent is used, however, the alloys are slightly stronger in tension, but become brittle. Hadfield, whose alloys contained about 0.2 per cent carbon, found the limit of forgeability between 5.6 and 9.14 per cent aluminum content, while Guillet found the limit between 7.18 and 9.25 per cent for alloys containing 0.8 per cent carbon. The results by these three investigators are shown graphically in Fig. 6.

The effect of aluminum on the magnetic properties of iron was studied to some extent in the early nineties [*] after the beneficial effect of this element as a purifier of iron and steel had been definitely established. It remained for Hadfield,[†] however, to make the first iron-aluminum alloys which had better magnetic qualities than those of the purest iron obtainable at the time. His 2.25 and 5.5 per cent aluminum alloys as tested by Barrett showed a higher permeability and a lower hysteresis loss than did the purest Swedish charcoal iron (see Table 1).

---

*Journal Iron and Steel Inst., vol. 38, II, p. 170, 1890.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

‡Oesterreich Z., fur das Berg—Hütten—und Salinenwesen, vol. 41, p. 536, 1893.

§Rev. de Metal. Memoirs, vol. 2, pp. 312-327, 1905. Journal Iron and Steel Inst., vol. 70, II, p. 16, 1906.

TABLE 1

MAGNETIC AND ELECTRICAL PROPERTIES OF HADFIELD'S IRON-ALUMINUM ALLOYS

MARK	Al	Max. Permeability	Permeability for H = 8	Max. Induction for H = 45	Retentivity in Terms of B	Coercive Force in Terms of H	Hysteresis loss Ergs per c. c. per cycle		Specific Electrical Resistance Microhms
							for H = 45	for B = 9000	
S.C.I.	.00	4 000	1 700	......	10 800	1.10	abt. 8 500	2 334	10.2
B	.00	.....	1 625	16 800	9 770	1.66	10 760	.....	10.9
1167 D	.75	.....	1 500	16 000	10 500	1.80	11 000	.....	22.0
1167 H	2.25	6 000	1 700	16 900	10 500	1.00	8 000	1 443	39.0
1167 I	5.50	.....	1 200	13 000	4 150	1.00	6 500	.....	70.0

From the results shown in the table and those previously obtained with silicon,[‡] aluminum and silicon appear to have very similar effects upon the magnetic properties of iron.

Dillner and Engstrom [§] have investigated the effect of aluminum and silicon on the magnetic properties of sheet iron as well as of castings. They found that silicon reduced the permeability and increased the hysteresis loss in sheets, but not in castings, and that aluminum had the opposite effect. Furthermore, it was found that a combination of silicon and aluminum gave the best results for sheets.

The main objection to employing aluminum formerly was its high cost. While silicon, in the form of ferrosilicon, could be obtained at a cost of 12 cents per pound of the element, the cost of aluminum was nearly $2 per pound. On this account silicon was employed in making steel for magnetic purposes, and, having given universal satisfaction both with regard to the hysteresis loss and to aging, it has retained its place. In spite of the reduced cost of aluminum, there has appeared to be no decided advantage in changing from silicon steel to aluminum steel. Investigators have turned their attention toward improving and perfecting silicon steel while comparatively little investigational work has been recorded regarding iron-aluminum alloys, particularly regarding their magnetic properties. It is hoped, therefore, that the present investigation will prove to be of interest not only on account of the vacuum method employed, but also because it is one of the few systematic investigations on the subject.

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[*] Transactions Am. Inst. Elect. Engrs., vol. 9, pp. 250-262 (and Discussion). The Electrician, vol. 29, p. 475, 1893.

[†] Scientific Trans. Royal Dublin Soc., vol. 7, ser. 2, pp. 67-126. Jan., 1900. Journal Inst. Elect. Engrs., vol. 31, pp. 674-729, 1902.

[‡] Scientific Trans. Royal Dublin Soc., vol. 7, ser. 2, pp. 27-126, Jan., 1900.

[§] Journal Iron and Steel Inst., vol. 67, I, pp. 474-480, 1905.

II. MATERIAL, APPARATUS, AND METHODS *

The iron used as the basis of the investigation was doubly refined electrolytic iron containing 0.01 per cent carbon or less and about 0.01 per cent silicon. It was cleaned with HCl, distilled water, and alcohol, and then dried by means of ether in vacuo.

The aluminum used as the alloying material analyzed as follows:

Si	0.20	per cent
Fe	0.17	" "
C	nil	
Al (by diff.)	99.63	" "

The melting was done in magnesia crucibles in an Arsem type vacuum furnace capable of melting 600 grams of iron; the pressure was less than 1 mm. of mercury. Attempts were first made to melt the iron and the aluminum together, but this method had to be given up for two reasons. First, the melting point of aluminum is so low compared with that of iron, that a considerable portion of aluminum evaporated before the iron was in a condition to combine with it. Furthermore, the aluminum vapor interfered with the operation of the furnace in such way that the voltage could not be maintained at a sufficiently high value to melt the iron completely. The result was that the Al_2O_3 formed, instead of rising to the surface, became entangled in the iron, often completely enclosing pieces of iron and resulting in imperfect ingots. Second, as will be shown, on account of the power of aluminum at high temperatures to reduce CO gas, this method of melting was certain to introduce more or less

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*For further information see "Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914; and "Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

carbon into the alloys. It became necessary, therefore, first to melt the iron and then, towards the end of the operation, to drop in the aluminum. This was accomplished without opening the furnace by suspending the aluminum (in the form of a wire or rod) from a very fine wire extended between insulated binding posts which passed through the cover of the furnace. In order to drop the aluminum, the fine wire was fused by connecting the binding posts for a moment to the terminals of the furnace. This method made it necessary to use a cover with a hole in it over the crucible, together with a funnel to direct the rod into the crucible.

It has already been mentioned as a well established fact that aluminum at high temperatures, when added to molten iron for instance, will reduce CO gas and thus liberate free carbon. That it will do so even at lower temperatures has been shown by Stead,[*] and in the present investigation this has been further confirmed.

It is evident that whatever gases remain in the vacuum furnace must consist largely of carbon monoxide. During the first trial experiments with the aluminum suspended from the top of the furnace, it was noticed, upon examining the aluminum after the operation, that the lower end of the wire was coated with a grayish substance. This substance was evidently the mixture of carbon and Al_2O_3 mentioned by Stead. The lower the wire reached, and consequently, the hotter it became, the larger the portion of it which became coated. In order to prevent the wire from becoming hot, it was made as short as possible and, during the later part of the investigation, was protected from direct radiation by means of an iron tube extending down into the furnace. But even with these precautions the tip of the wire or rod was always slightly coated, particularly when more than 5 per cent aluminum was used. Moreover, when the aluminum was dropped in, it obviously became hot and may have had time in a few cases to reduce some CO. In these cases a small amount of carbon was invariably introduced into the alloy, generally not exceeding 0.02 per cent, but in some cases, especially during the early part of the investigation, amounting to 0.10 per cent or even more. One alloy analyzed as high as 0.65 per cent carbon.

It is quite evident from this discussion that in the type of furnace used it would be out of the question to mix the iron and the aluminum, melt them together, and expect to obtain an alloy with

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[*] Journal Iron and Steel Inst., vol. 38, II, p. 190, 1890.

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a low carbon content. At first this method was used without bad effects for very low percentages of aluminum when the iron oxide present evidently was sufficient to oxidize both the aluminum and the carbon, but the method was abandoned for aluminum contents above 0.5 per cent, for the reasons already stated.

The alloys known or suspected to contain more than 0.015 per cent carbon for low aluminum alloys and more than 0.025 per cent carbon for high aluminum alloys have been tabulated separately in order to show the effect of carbon upon the various properties. It appears that the methods employed were not ideal, and that small amounts of carbon may have been introduced with the aluminum. It is possible, therefore, that better results, magnetically, might have been secured had a furnace been employed, having, for instance, a tungsten heating element.

The ingots were allowed to cool while in vacuo, and then were forged into rods and machined into test pieces.

The magnetic testing was done by means of the Burrows compensated double bar and yoke method. It has been previously shown* that the errors made by using this method for very high permeability iron may be much greater than theoretical considerations would indicate, but no corrections have been made in the present case, it being desired that the results given in this bulletin be comparable with those given in the previous bulletin on iron-silicon alloys. Furthermore, the question of corrections is a complicated one and has by no means been definitely settled, for the amount of compensating current needed to equalize the flux along the rod depends largely upon the uniformity of the rods used. This question is further discussed in the Appendix. The Appendix contains also additional data obtained from ring specimens, showing that for permeabilities not exceeding 20,000, the results obtained with the Burrows method may be relied upon as being very close to the correct values.†

The tests for electrical resistance were made by means of a Kelvin double bridge, using standardized iron rods for comparison.‡ For the mechanical testing an Olsen 10,000 pound testing machine was employed, and the usual characteristics were obtained. The rods were tested magnetically and electrically after the following heat treatments:

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*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, pp. 24-27, 1915.

†"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, Appendix, 1915.

‡The standardization was done by the Leeds and Northrup Co.

a. As forged
b. Annealed at 900 degrees C. Cooled at the rate of 30 degrees per hour
c. Annealed at 1,100 degrees C. Cooled at the rate of 30 degrees per hour

Mechanical tests were made on unannealed specimens and on specimens heated to 1,000 degrees C. and then cooled at the rate of 30 degrees C. per hour. The heat treatment was applied in vacuo in order to prevent oxidation, a vacuum of 0.2 to 0.1 mm. being maintained during the treatment. The specimens were packed in powdered magnesia in an electroquartz tube.

III. CHEMICAL ANALYSIS

The marked effect of aluminum and silicon upon the electrical resistance of iron is well known. One per cent of either element added to iron will more than double its electrical resistance, and the increase is nearly proportional to the amount added. Furthermore, the resistance is not appreciably affected by mechanical or thermal treatments. These facts furnish an excellent opportunity for establishing the aluminum content of iron which contains only small amounts of impurities. This opportunity has been used to advantage in the present case. A few alloys were selected the aluminum content of which, as determined approximately by the aluminum added to the iron, covered the range studied. After the magnetic and electrical tests on these alloys were completed, the test rods were placed in a lathe, the surface layer was removed, and fine shavings were collected from along the rods in sufficient quantities for duplicate chemical analysis.

The results of this analysis * are given in Table 2, Column 3; from these figures and those for the electrical resistance, given in Column 5, the curve in Fig. 2 showing the relation between the resistance and the aluminum content was established. The aluminum content of all the other alloys for which electrical resistance measurements could be made, was then determined by means of the curve in Fig. 2. The values thus obtained have been checked by further analysis and found to be within the limits of accuracy desired.

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*The method used is described by Mr. J. M. Lindgren in the Appendix.

It was stated in the previous chapter that, due to the conditions of melting and to the reduction of CO by aluminum, a large number of the alloys contained carbon. Therefore, a careful weeding of the alloys was made.

TABLE 2

LIST OF ALLOYS AND THEIR CHEMICAL ANALYSIS

No.	Aluminum Content		Carbon Content Per Cent by Chem. Analysis	Specific Elec. Resistance Annealed at 1100° C. Microhms	Aluminum Content from Res. Curve Per Cent	Remarks
	Per Cent As Charged	Per Cent by Chem. Analysis				
3 Al. 05	0.1	0	....	10.2	0.02	Aluminum mixed with iron before melting.
06	0.2	0	....	10.4	0.05	
07	0.3	Trace	....	10.4	0.05	
08	0.4	Trace	....	9.9	0.00	
09	0.5	0.09	0.05	11.0	0.09	Classed as contaminated.
10	0.6	0.11	....	12.0	0.18	
11	0.7	0.09	....	10.9	0.08	
12	1.0	....	....	14.4	0.40	
18	....	0.11	....	12.6	0.23	
19	0.0	....	....	9.7	0.00	
20	1.0	....	0.01	19.2	0.80	
21	2.0	....	0.01	30.2	1.80	
22	....	....	0.02	15.3	0.45	Classed as contaminated.
23	3.0	....	0.01	39.7	2.67	
24	4.0	3.53	0.06	49.0	3.58	Classed as contaminated.
25	5.0	4.83	0.16	57.7	4.52	
26	....	7.63	0.45	....	....	
27	7.6	8.10	0.09	81.4	7.95	
28	....	12.36	0.65	....	....	
29	....	13.05	0.38	....	....	
30	0.2	....	....	11.0	0.09	
31	0.4	....	....	11.1	0.09	
32	0.6	....	....	11.6	0.10	
33	0.8	....	....	14.1	0.37	
34	1.0	....	0.02	12.4	0.22	Classed as contaminated. Only small portion of added Al. apparently dissolved in the iron. Consequently carbon believed to be high.
35	1.5	....	....	15.0	0.44	
36	2.0	0.73	....	18.0	0.69	
37	2.5	....	....	19.0	0.78	
39	3.0	....	....	13.7	0.32	
40	3.4	3.31	....	43.9	3.08	
41	3.9	....	....	57.8	4.54	
42	4.4	....	....	51.2	3.82	
43	4.8	....	....	57.9	4.55	
45	5.7	....	....	63.5	5.20	
46	6.7	5.30	0.02	67.7	5.72	
47	....	8.60	0.13	84.3	8.75	Classed as contaminated.
49	6.6	....	....	70.4	6.24	
50	....	1.55	....	29.4	1.70	
51	7.7	....	0.11	66.1	5.53	Classed as contaminated.

The alloys obtained by mixing the aluminum with the iron before melting includes Nos. 3 Al 05-17. Of these, only Nos. Al 05-08 were retained as being uncontaminated; the ingots for Alloys Nos. 3 Al 13-17 were decidedly unsound. Those in the next series, 3 Al 18-29, inclusive, were prepared by dropping the aluminum into the molten iron. Most of the alloys of this series contain large amounts of carbon, for the aluminum was not protected from undue heating.

Furthermore, when a large piece of aluminum was dropped into the crucible, such agitation resulted that some of the molten iron undoubtedly came in contact with the small graphite funnel, which was used to guide the aluminum into the crucible. Finally, during the melting of the Alloys 3 Al 26, 28, and 29, the MgO crucible cracked and the iron came in contact with the graphite container. In the last series, 3 Al 30-51, inclusive, an alundum or magnesia funnel was used, and the aluminum was protected from undue heating by means of an iron tube. In this series, 3 Al 34-39, inclusive, were classed as contaminated because the aluminum content of the iron was, for

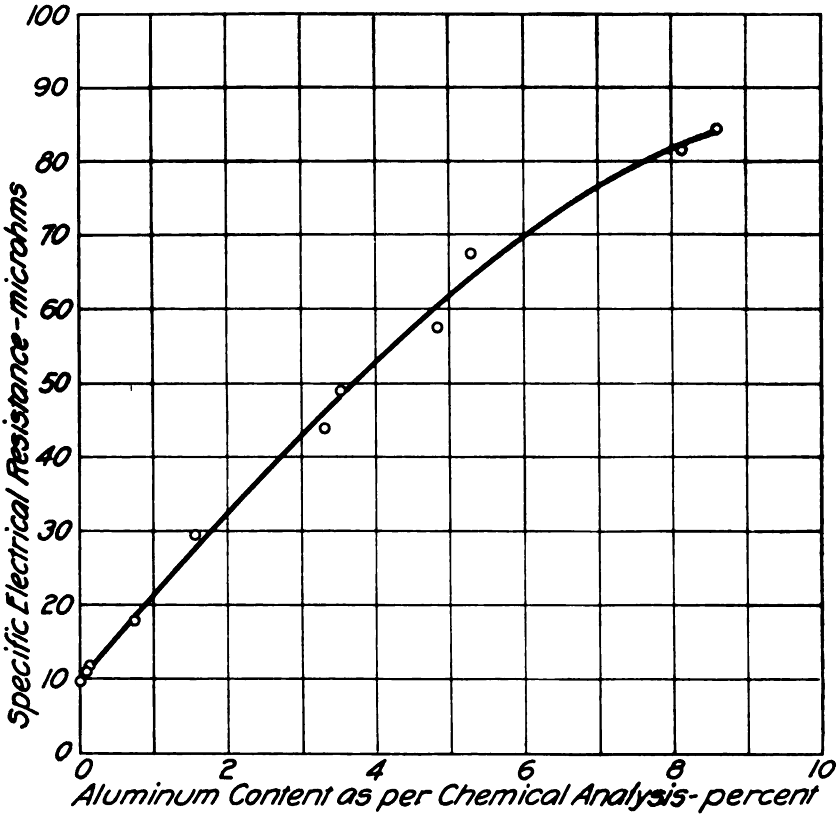

FIG. 2. ELECTRICAL RESISTANCE OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C.

TABLE 3

RESULTS OF MECHANICAL TESTS. AS FORGED

Spec. No.	Aluminum Content Per Cent	Carbon Content Per Cent	Yield Point lbs. per sq. in.	Ult. Str. lbs. per sq. in.	Elongation Per Cent		Reduction of Area Per Cent	Remarks
					Before Necking	Ultimate		
8 Al. 19	0.00	...	50700	54700	4.0	26.0	84.3	
05	0.02	...	40200	46400	5.0	28.0	93.4	
31	0.09	...	48400	46200	3.0	28.0	88.4	
20	0.80	.01	86300	86500	....	25.0	86.9	Broke outside punch marks.
50	1.70	...	79100	83400	3.0	3.0	6.8	
21	1.80	.01	63600	64800	3.0	20.0	82.8	
23	2.67	.01	47700	59700	13.0	36.0	81.6	Broke outside punch marks.
40	3.08	...	68200	77500	4.0	21.0	76.4	
42	3.82	...	73400	77800	3.0	25.0	82.7	
41	4.54	...	57600	66300	8.0	28.0	82.6	
43	4.55	...	81800	84400	2.5	23.0	81.6	
45	5.20	...	80000	82800	1.0	20.0	84.3	
46	5.72	.02	78400	86700	5.0	26.0	82.6	
49	6.24	...	77700	86000	5.0	28.0	74.7	

CONTAMINATED ALLOYS

8 Al. 11	0.08	...	39200	46360	11.0	33.0	89.8	
34	0.22	.02	58600	64400	1.0	12.0	55.0	
39	0.32	...	84800	88700	3.0	22.0	79.6	
35	0.44	...	62000	72100	6.0	22.0	73.5	
36	0.69	...	77700	83300	8.0	25.0	82.5	
37	0.78	...	79700	84900	3.0	20.0	73.6	Broke outside punch marks.
24	3.58	.06	61400	75300	7.0	25.0	76.8	Broke outside punch marks.
25	4.52	.16	90800	96600	1.0	1.0	2.5	
51	5.53	.11	89100	96600	0.0	0.5	2.1	
26	7.63	.45	71900	71900	1.0	1.0	0.5	
27	7.95	.09	72800	84100	11.0	27.0	76.6	
47	8.75	.13	64900	73600	5.0	18.0	36.3	
28	1312.6	.65	85200	90500	1.5	1.5	7.5	Broke outside punch marks.
29	03.5	.38	96500	115000	0.0	0.5	2.6	

TABLE 4

RESULTS OF MECHANICAL TESTS. ANNEALED AT 1000 DEGREES C.

Spec. No.	Aluminum Content Per Cent	Carbon Content Per Cent	Yield Point lbs. per sq. in.	Ult. Str. lbs. per sq. in.	Elongation Per Cent		Reduction of Area Per Cent	Remarks
					Before Necking	Ultimate		
3 Al. 19	0.00	....	17600	34900	33	60	93.5	
08	0.00	....	14000	32100	29	48	65.7	
05	0.02	....	13900	34470	32	60	91.6	
30	0.09	....	13800	36400	27	50	84.3	
31	0.09	....	13700	33700	28	62	89.0	
32	0.10	....	14200	35900	31	57	91.6	
20	0.80	0.01	21700	40100	26	56	91.5	Flaw in Test piece.
21	1.80	0.01	23900	46000	27	61	90.7	
23	2.67	0.01	30100	49400	28	49	89.0	
40	3.08	....	31800	53400	25	51	85.3	
42	3.82	....	37300	55900	24	51	85.1	
41	4.54	....	18100	43800	21	54	85.9	Broke at punch mark. Data not plotted.
43	4.55	....	41900	60200	22	49	83.8	
45	5.20	....	45000	61600	20	43	83.8	
46	5.72	0.02	50200	67400	18	40	79.7	
49	6.24	....	53400	69800	11	27	55.5	

CONTAMINATED ALLOYS

3 Al. 09	0.09	0.05	21000	37900	33	60	85.1	
10	0.18	....	18800	37100	24	53	79.8	
34	0.22	0.02	13500	37700	25	51	83.0	
39	0.32	....	50700	67500	14	32	79.2	
35	0.44	....	21000	46800	28	50	83.8	
36	0.69	....	34900	55300	24	49	83.0	
37	0.78	....	38300	56800	24	48	86.9	
24	3.58	0.06	39500	56900	27	54	76.8	
25	4.52	0.16	39200	60800	22	46	76.9	
51	5.53	0.11	79600	84800	1	1	2.2	
26	7.63	0.45	58400	63200	3	3	2.2	Broke outside punch mark.
27	7.95	0.09	26100	50300	20	45	89.5	
47	8.75	0.13	24900	50300	21	50	85.6	
28	12.36	0.65	26100	59000	4	4	6.1	
29	13.05	0.38	91500	91500	1	1	0.0	Broke outside punch mark.

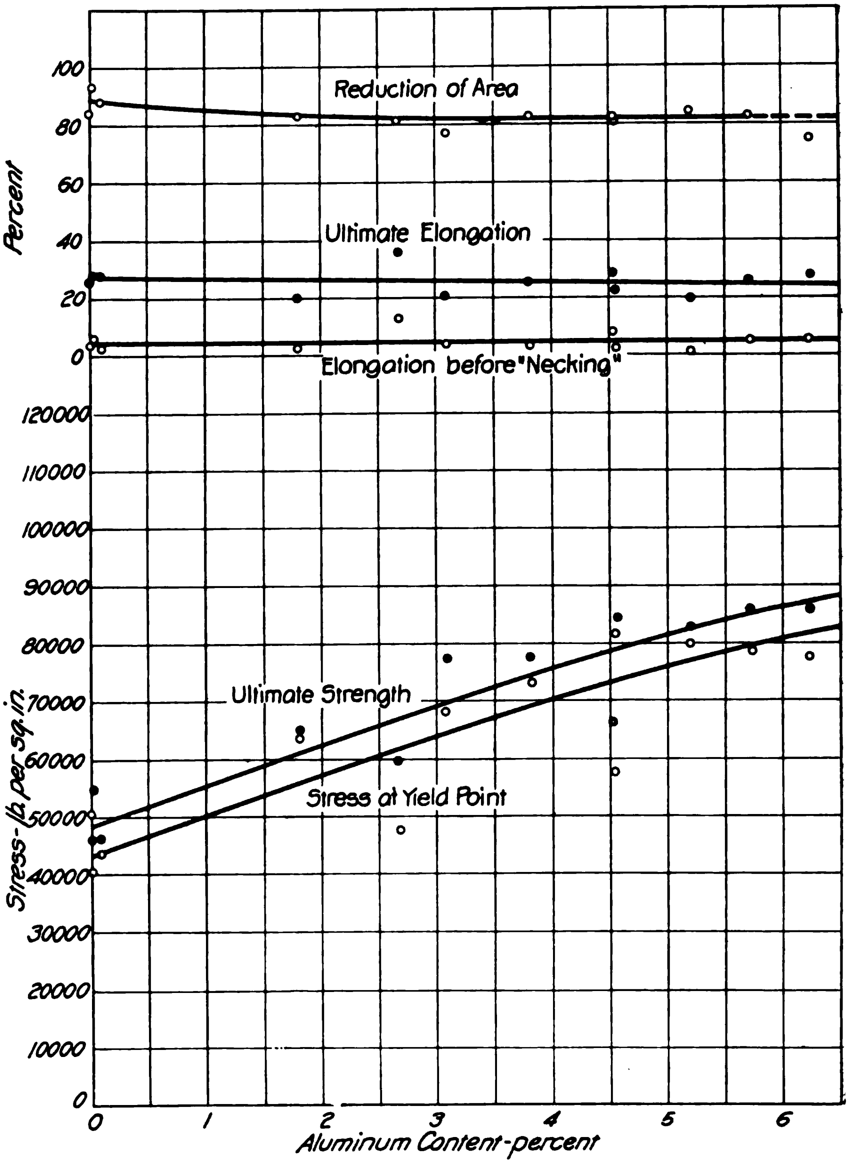

FIG. 3. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. AS FORGED

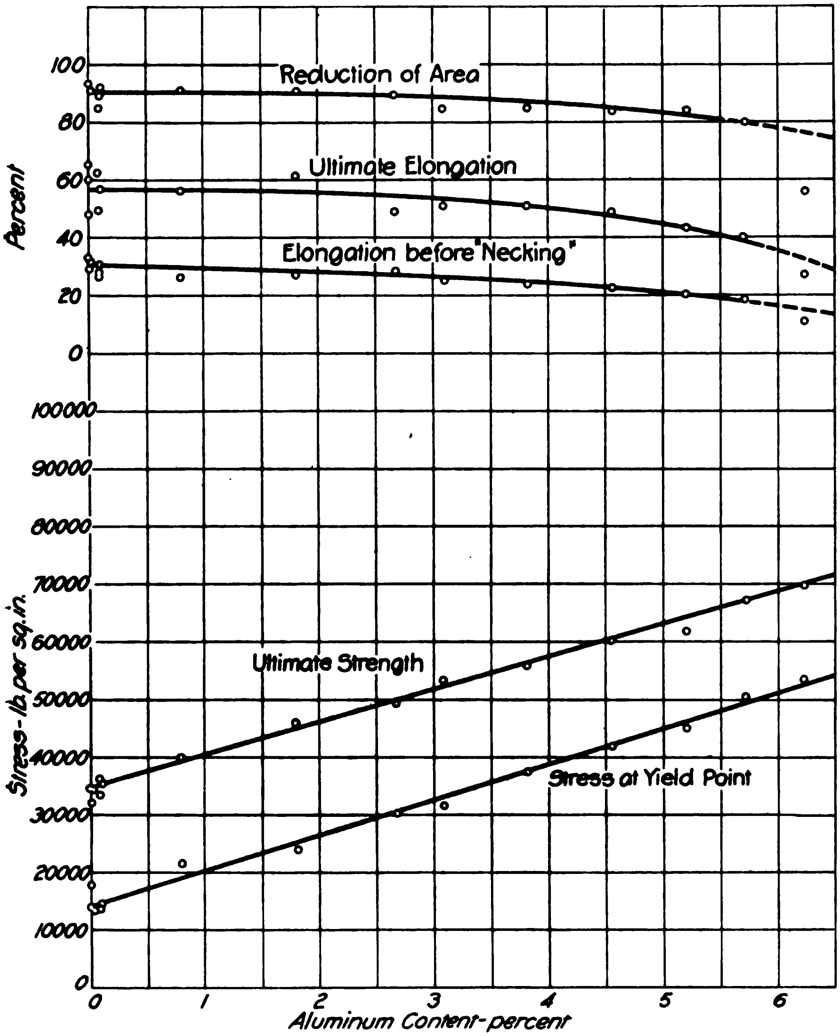

FIG. 4. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1000 DEGREES C.

some unaccountable reason, too low on comparison with the aluminum added. If the carbon introduced is in proportion to the aluminum added, then the carbon content of these alloys was quite out of proportion to the aluminum content.

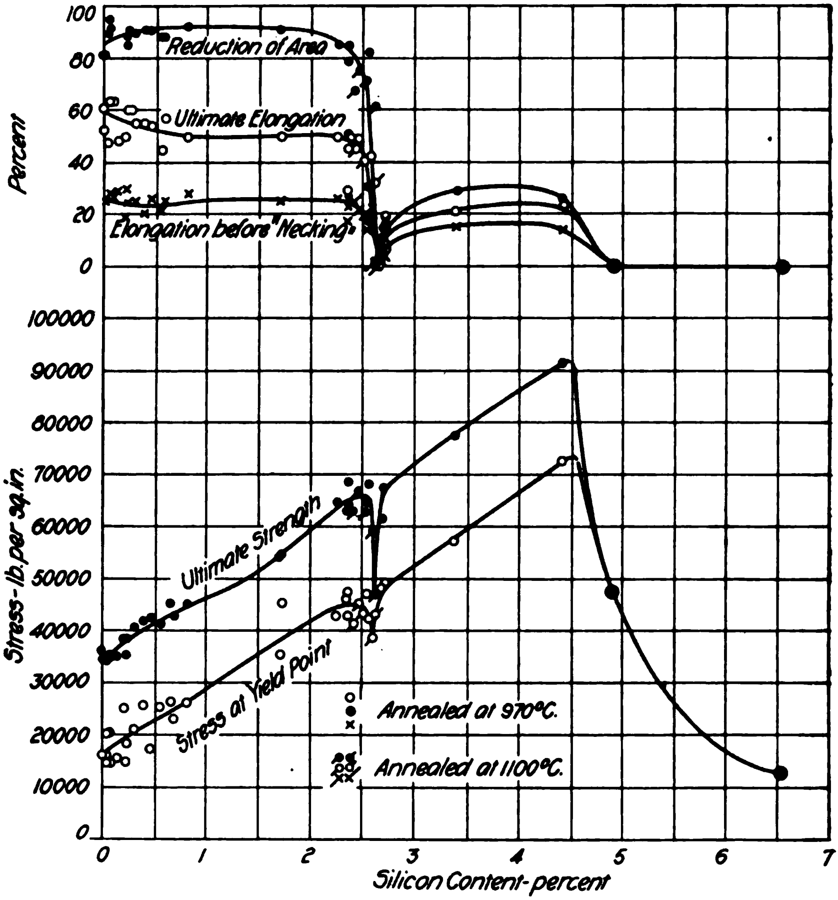

FIG. 5. MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS MELTED IN VACUO. ANNEALED
(FROM BUL. 83)

As a check on this weeding, a number of these alloys were analyzed for carbon; the results of this analysis are shown in Column 4 of Table 2. For low aluminum alloys a carbon content of 0.015 per cent has been allowed; for high aluminum alloys, 0.025 per cent.

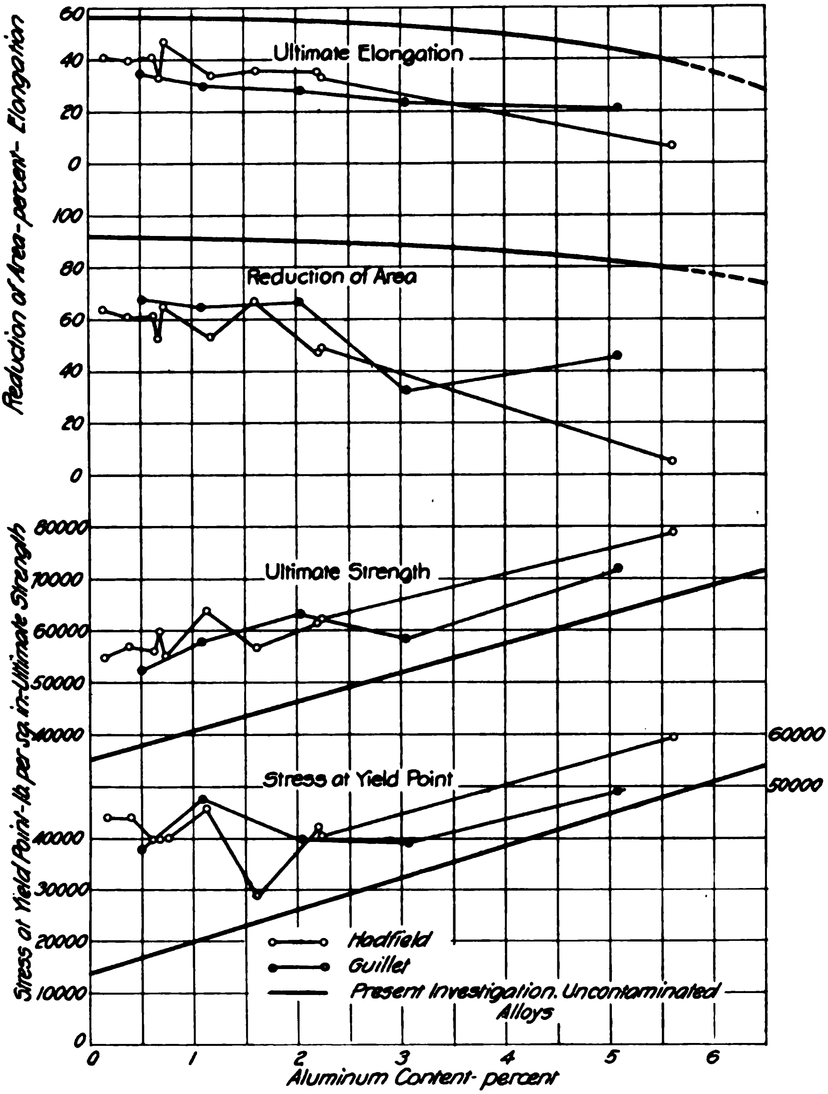

FIG. 6. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED

IV. RESULTS

1. *Mechanical Properties.*—The results of the mechanical tests are shown in Tables 3 and 4 and in Figs. 3, 4, 7, and 8. For the

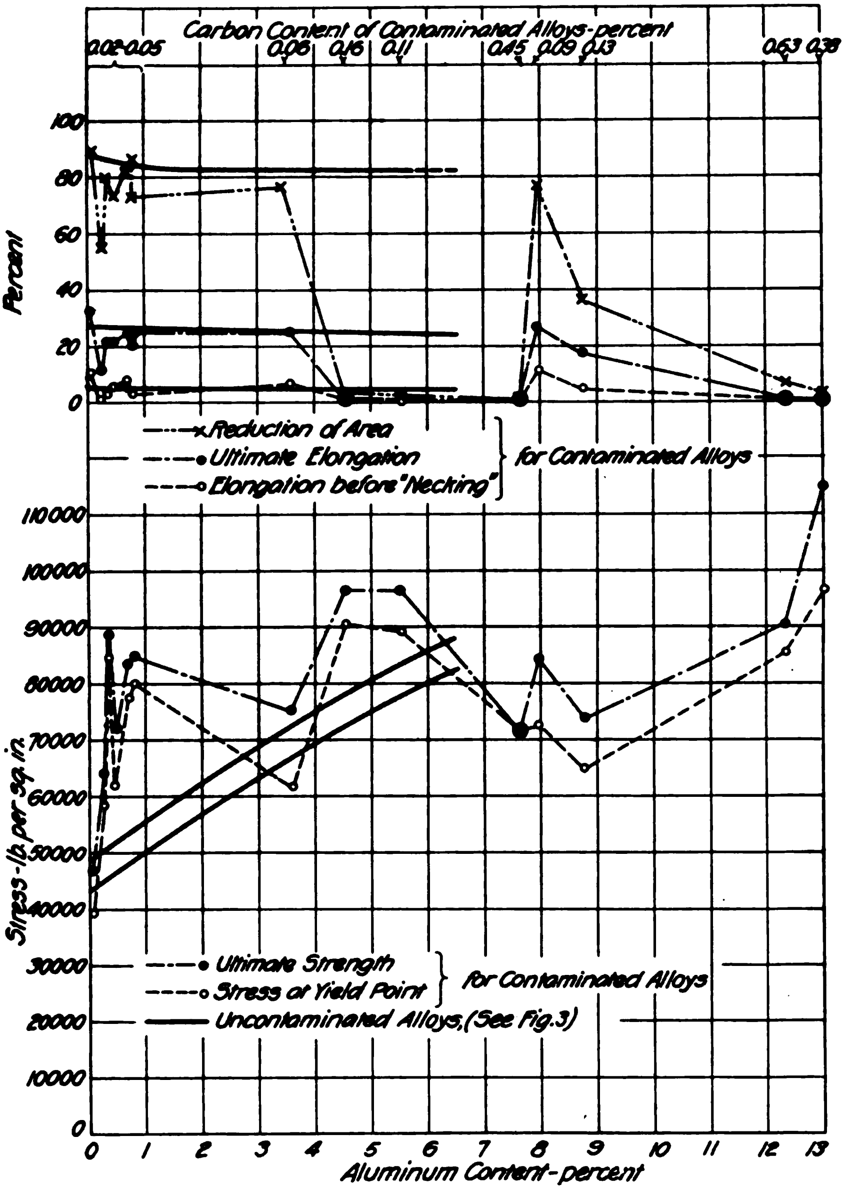

FIG. 7. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. AS FORGED

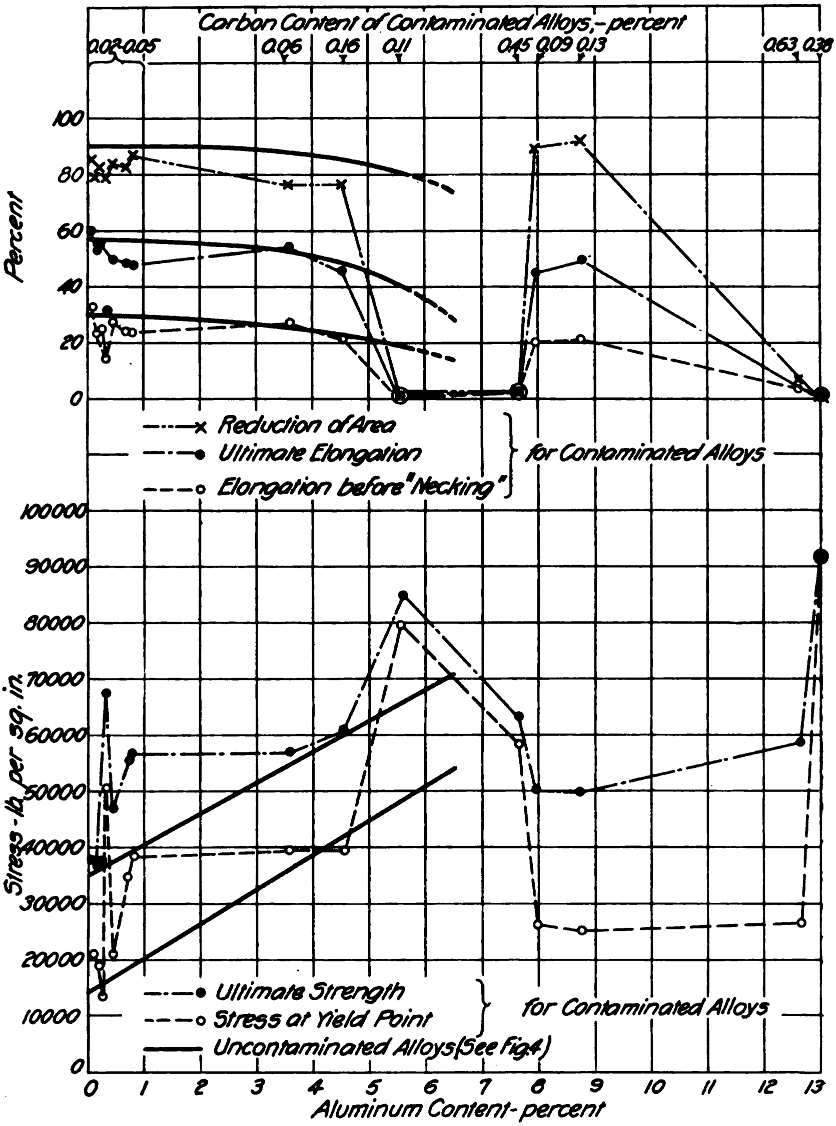

FIG. 8. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. ANNEALED AT 1000 DEGREES C.

sake of comparison, the mechanical properties of the corresponding iron-silicon alloys are shown in Fig. 5.* In Fig. 6 the results of the present investigation are compared with those obtained by Hadfield† and Guillet.‡ Fig. 9 is a photograph of some of the mechanical test pieces after testing.

From the data thus presented, it may be stated in general that aluminum in the absence of carbon increases the strength of iron in almost direct proportion to the amount added. Furthermore, aluminum appears to affect the toughness of iron only slightly. To what aluminum content these rules can be applied is somewhat uncertain, as no alloy containing more than about 6 per cent aluminum uncontaminated by carbon was obtained. How carbon affects these properties may be judged from Figs. 7 and 8. It is apparent that for alloys containing in the neighborhood of 0.1 per cent carbon, aluminum, up to at least 8 per cent, has no marked effect either upon the strength or upon the toughness of iron. This is especially evident in the case of the annealed alloys. The effect of small amounts of carbon is particularly great upon nearly pure iron; it increases the strength of iron about 50 per cent. The strength curves for alloys containing small amounts of carbon will, consequently, be nearly horizontal, and will cross the curves for the more nearly carbon-free alloys at 4 to 5 per cent aluminum, where the effect of carbon seems to be very small. In general the effect of small amounts of carbon is to conceal the true effect of aluminum upon the mechanical properties of iron. If the results given here for alloys with a carbon content of about 0.1 per cent are compared with those given by Hadfield and Guillet (see Fig. 6), they will be found to be very similar. They are more like Guillet's results, which represent alloys containing about 0.1 per cent carbon, than Hadfield's, whose alloys contained about 0.2 per cent. For carbon contents of 0.4 to 0.6 per cent the alloys are forgeable up to at least 13 per cent aluminum. This also agrees with the results of Guillet. For such carbon contents, however, the alloys are quite brittle.

If the effect of aluminum (Fig. 4) upon the mechanical properties is compared with the effect of silicon (Fig. 5), it will be seen that, up to 4.5 per cent, silicon increases the strength much more than does aluminum. On the other hand, although 2.5 to 4.5 per cent silicon

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*Taken from "Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

‡Ibid., vol. 70; II. p. 16, 1906. Rev. de Metal. Memoirs. vol. 2, pp. 312-327. 1905.

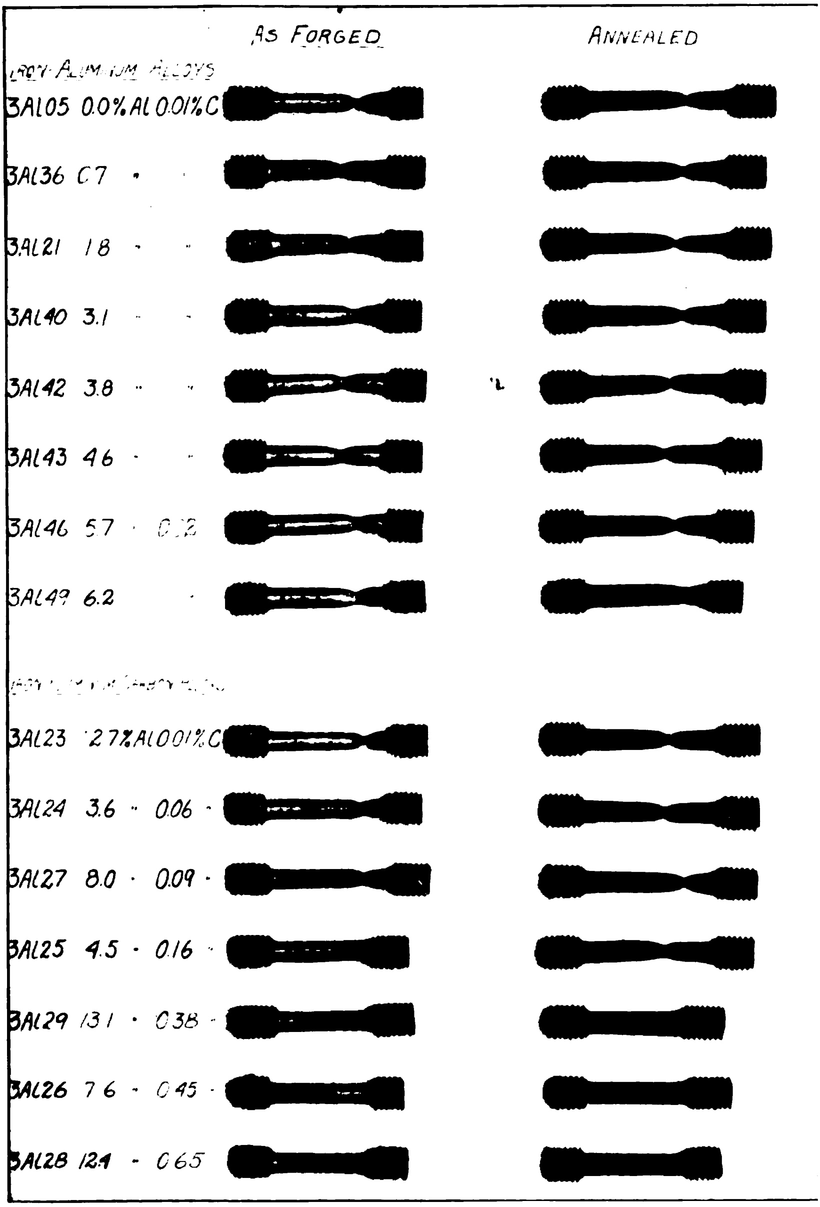

FIG. 9. SOME OF THE MECHANICAL SPECIMENS AFTER BEING TESTED

markedly increases the brittleness of iron, aluminum has no such effect. Furthermore, silicon beyond 4.5 per cent rapidly decreases both the strength and the toughness, while aluminum continues to add strength to the iron without materially affecting the toughness.

2. *Magnetic and Electrical Properties.*—The results of the magnetic and electrical test are shown in Tables 5 to 7 and in Figs.

TABLE 5

RESULTS OF MAGNETIC AND ELECTRICAL TESTS

RODS UNANNEALED

Rod No.	Aluminum Content Per Cent	Carbon Content Per Cent	Maximum Permeability	Density for Max. Permeability Gausses	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gausses		Coercive Force Gilberts per cm.		Spec. Elec. Res. at 20° C. Microhms
					For B = 10000	For B = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	
3Al.19	0.00	....	1500	6000	1162	605	5340	10100	5080	6560	1.7	2.1	9.7
08	0.00	....	3460	4500	2670	1455	....	....	....	....	...	...	....
05	0.02	....	2260	5200	1885	1035	....	....	....	....	...	...	....
06	0.05	....	3120	5000	2100	1052	....	....	....	....	...	...	....
07	0.05	....	3570	5000	2500	1315	....	....	....	....	...	...	....
30	0.09	....	1935	6000	1510	824	5230	9920	6400	7100	1.9	2.1	11.1
31	0.09	....	2180	4800	1665	770	4200	6620	6000	6400	1.5	1.6	11.2
32	0.10	....	1700	6800	1370	605	5850	10910	6260	7300	2.3	2.35	11.5
33	0.37	....	1650	6600	1280	564	5770	10550	6200	6800	2.0	2.2	14.3
20	0.80	0.01	1620	6500	1250	577	5890	10630	6580	6980	2.1	2.4	19.4
50	1.70	....	2240	5600	1430	527	....	....	....	....	...	...	....
21	1.80	0.01	967	6000	770	429	8460	14600	5900	6560	3.1	3.3	31.0
23	2.67	0.01	2400	4820	760	394	5080	8970	4770	4870	1.9	2.1	40.2
40	3.08	....	1067	4800	769	366	7690	12840	5100	5500	3.0	3.2	44.6
42	3.82	....	880	5300	555	263	8580	15500	4850	5000	3.5	3.7	52.0
41	4.54	....	688	4200	435	187	11950	19450	5400	5800	4.2	4.6	58.0
43	4.55	....	763	4200	477	220	9600	16000	4500	4700	3.2	3.5	59.0
45	5.20	....	616	4000	357	168	11700	17100	4500	4600	4.3	4.8	64.0
46	5.72	0.02	585	4000	320	150	9860	14500	4000	4150	3.7	3.7	68.5
49	6.24	....	600	3200	333	160	....	....	....	....	...	...	....
CONTAMINATED ALLOYS													
11	0.08	....	2170	5000	1665	926	....	....	....	....	...	...	....
09	0.09	0.05	2410	4820	1850	943	....	....	....	....	...	...	....
10	0.18	....	2000	5600	1560	823	....	....	....	....	...	...	....
34	0.22	0.02	2000	6000	1500	528	4400	7630	6200	6700	1.4	1.6	12.7
18	0.23	0.11	1500	6000	1086	550	6480	9600	6290	6760	2.3	2.3	12.6
39	0.32	....	2280	6600	1886	728	4450	7750	4400	4800	1.5	1.6	14.3
12	0.40	....	1400	7000	1075	450	7450	14000	6800	7800	2.6	3.1	14.4
35	0.44	....	2140	6000	1820	833	5090	8400	7800	8200	1.5	1.9	15.3
22	0.45	0.02	1590	6600	1390	667	6360	11000	7105	8160	2.2	2.6	16.1
36	0.69	....	1930	5600	1430	625	4900	10500	5800	7400	1.7	2.0	18.1
37	0.78	....	1870	5600	1470	750	4960	8900	6400	7000	1.7	1.9	19.5
24	3.58	0.06	1070	4600	595	281	8270	14000	5300	5300	2.7	2.7	49.5
25	4.52	0.16	573	4300	384	200	12700	21900	5000	5600	4.5	5.1	57.5
51	5.53	0.11	770	3100	370	150	....	....	....	....	...	...	...
27	7.95	0.09	450	3600	202	104	11450	20400	4000	4400	4.7	4.9	78.2
47	8.75	0.13	500	4000	...	...	....	....	....	....	...	...	...

TABLE 6

RESULTS OF MAGNETIC AND ELECTRICAL TESTS ANNEALED AT 900 DEGREES C.

Rod No.	Aluminum Content Per Cent	Carbon Content Per Cent	Maximum Permeability	Density for Max. Permeability Gausses	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gausses		Coercive Force Gilberts per cm.		Spec. Elec. Res. at 20° C. Microhms
					For B = 10000	For B = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	
3Al.19	0.00	....	17800	9000	17750	8290	954	2060	9080	14200	.33	.40	9.7
08	0.00	....	30000	12000	29400	25000	795	1470	9400	14600	.27	.30	9.8
05	0.02	....	24300	12160	22250	12500	986	2240	9360	14600	.32	.43	10.2
06	0.05	....	18500	10000	18500	9100	985	2060	9120	14080	.33	.42	10.5
07	0.05	....	18300	7700	16800	10000	750	1715	8800	12180	.23	.38	10.3
30	0.09	....	17280	8700	16650	6520	1037	2455	9200	13700	.33	.48	11.4
31	0.09	....	19500	6400	14500	3840	795	1825	9200	11800	.24	.29	11.2
32	0.10	....	22500	9000	20800	3950	840	2240	9200	13400	.27	.42	11.2
33	0.37	....	23500	8000	20400	3750	827	2085	9200	13700	.27	.38	14.1
20	0.80	0.01	14000	8000	12500	3950	878	2458	8600	12750	.35	.47	19.2
21	1.80	0.01	10700	11000	9170	3340	1650	3240	9195	12870	.52	.63	30.7
23	2.67	0.01	12300	8000	11750	1250	1195	2680	8600	11700	.40	.60	39.9
40	3.08	....	13800	8000	12500	1085	1175	2670	9100	12800	.39	.47	44.3
42	3.82	....	12700	7000	10500	600	1157	2920	8700	11300	.36	.43	51.6
41	4.54	....	12360	6800	9270	593	1273	2860	9000	11100	.41	.47	57.6
43	4.55	....	14600	7200	12560	590	1304	3440	9600	11900	.42	.57	57.3
45	5.20	....	14680	7200	11500	428	1170	3200	9100	11000	.38	.45	63.0
46	5.72	0.02	11500	6000	7700	273	1290	3150	8600	10200	.41	.45	68.7

CONTAMINATED ALLOYS

11	0.08	....	13700	8200	13300	6770	1113	2670	8800	13800	.37	.52	11.0
09	0.09	0.05	10750	8600	10500	4840	1227	2740	8520	12200	.37	.50	11.1
10	0.18	....	11850	9000	11750	5000	1270	2630	8800	12500	.42	.47	12.1
34	0.22	0.02	12000	6000	9100	1610	1080	2730	8300	10600	.33	.43	12.6
39	0.32	....	20000	10000	20000	6380	902	1905	9400	13200	.30	.40	13.6
35	0.44	....	15800	10000	14700	5000	1233	2900	9200	14400	.42	.58	15.1
22	0.45	0.02	6500	8000	6060	2140	1930	4590	7990	11380	.35	.45	15.6
36	0.69	....	18000	9000	17550	3950	985	2170	9100	13700	.33	.41	17.5
37	0.78	....	12500	7000	11100	3750	1227	2605	8900	12750	.38	.46	18.9
24	3.58	0.06	5630	5200	3570	600	1615	4293	5800	7400	.57	.73	49.2
25	4.52	0.16	2000	4000	1100	341	2490	5530	4300	4700	1.20	1.30	56.8
27	7.95	0.09	3800	3000	1000	150	2350	5080	4500	4550	.80	.80	86.1

TABLE 7

RESULTS OF MAGNETIC AND ELECTRICAL TESTS ANNEALED AT 1100 DEGREES C.

Rod No.	Aluminum Content Per Cent	Carbon Content Per Cent	Maximum Permeability	Density for Max. Permeability Gausses	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gausses		Coercive Force Gilberts per cm.		Spec. Elec. Res. at 20° C. Microhms
					For B = 10000	For B = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	
19	0.00	....	17500	7000	16650	7500	890	2100	8835	13000	.30	.41	9.7
08	0.00	....	22100	6630	20000	7130	700	1370	9400	12200	.22	.26	9.9
05	0.02	....	20800	6200	18150	7500	699	1780	9195	13000	.23	.35	10.2
06	0.05	....	20500	10300	20000	10100	780	1971	9200	13710	.26	.43	10.4
07	0.05	....	30150	9045	30150	12000	670	1620	9400	14200	.22	.33	10.4
30	0.09	....	23200	6800	17850	5000	610	1261	9075	11180	.19	.22	11.0
31	0.09	....	26000	5600	17250	4060	522	1080	8520	10200	.16	.19	11.1
32	0.10	....	34600	7600	30300	6530	427	985	9000	11800	.14	.16	11.6
33	0.37	....	42700	8400	37600	3750	445	1080	9300	13400	.14	.20	14.1
20	0.80	0.01	20900	7000	17230	3950	700	1590	8920	11800	.23	.25	19.2
50	1.70	....	22230	6500	18170	2210	670	1560	8900	11800	.21	.25	29.4
21	1.80	0.01	18500	8500	17100	3000	720	1755	9115	13270	.26	.30	30.2
23	2.67	0.01	19700	7500	16150	2000	763	1790	8800	11400	.23	.26	39.7
40	3.08	....	22100	8600	20000	1150	840	2670	9400	13100	.28	.36	43.9
42	3.82	....	14600	6000	13150	600	973	2290	8900	11300	.33	.41	51.2
41	4.54	....	14480	6800	11000	537	1100	2645	8800	10800	.37	.45	57.8
43	4.55	....	19800	6000	11750	483	985	2530	8800	11300	.27	.52	57.9
45	5.20	....	21760	6600	11500	375	793	2450	8900	10300	.25	.32	63.5
46	5.72	0.02	12450	6000	7400	273	1210	2300	8800	9900	.38	.43	67.7
49	6.24	....	16050	3200	7400	259	922	2830	8200	9100	.27	.31	70.4

CONTAMINATED ALLOYS

11	0.08	....	15000	6500	13900	6300	954	2105	9200	12400	.31	.40	10.9
09	0.09	0.05	11000	8800	10500	4970	1370	3080	9200	12990	.43	.63	11.0
10	0.18	....	11650	7000	10500	4170	1087	2860	8300	12100	.30	.44	12.0
34	0.22	0.02	18100	5400	11370	1630	706	1423	8000	8500	.20	.20	12.4
39	0.32	....	20100	7000	17550	7500	890	1843	9500	13000	.29	.35	13.7
35	0.44	....	16000	9000	15880	5775	1080	2280	9600	14200	.35	.42	15.0
22	0.45	0.02	10200	9000	10000	3000	1270	2960	8400	12590	.42	.52	15.3
36	0.69	....	20440	9000	20000	5000	1005	2220	9500	14400	.32	.37	18.0
37	0.78	....	15500	6000	13330	5000	1023	2100	9200	13000	.33	.36	19.0
24	3.58	0.06	6250	5000	3570	600	1390	3330	6100	6800	.45	.53	49.0
25	4.52	0.16	2160	4000	1110	341	2860	5520	4300	4500	1.10	1.20	57.7
51	5.53	0.11	8430	2500	2220	283	1335	3700	4900	5100	.36	.40	66.1
27	7.95	0.09	3060	2300	714	150	2065	2960	3140	3140	.80	.80	81.4

10, 11, 12, 14, and 15. In Fig. 13 the properties of the iron-silicon alloys annealed at 1100 degrees are given for comparison with the corresponding iron-aluminum alloys.

The specific electrical resistance unaffected by any heat treatment, which is about 9.8 microhms for pure iron, is seen to increase by 12 microhms for each per cent aluminum added up to an aluminum content of 3 per cent. For higher values the increase falls off gradually.

Although the electrical resistance is unaffected by heat treatments, the magnetic properties, as is well known, are extremely sensitive to them. From previous investigations on this subject,* it seems probable that the factor which determines the magnetic properties of a certain iron or iron alloy is the internal strain existing in the metal, and that it is the removal of this strain by annealing and slow cooling that is the real cause of any increase in permeability and decrease in hysteresis loss. The transformation of all the β- and γ-iron into α-iron is undoubtedly essential; but in dealing with pure iron, or with iron-aluminum or iron-silicon alloys, this transformation is supposed to take place quite readily, and does not necessitate as slow cooling as 30 degrees per hour. Almost any rate of cooling, short of quenching, seems to effect this transformation, though it has been shown repeatedly that as far as the magnetic properties are concerned, a cooling at the rate of 100 degrees per hour is much inferior to a rate of 30 degrees. A slower rate than 30 degrees per hour does not appear to be of any further advantage. Again, although the transformation from γ- to β-iron (if the β modification actually does exist) takes place at 900 degrees for pure iron, it has been shown that† the most noticeable increase in permeability for low and medium densities takes place upon annealing at between 700 and 800 degrees C. A further, but much smaller, increase takes place upon annealing at 900 degrees. Finally, it was shown in Bulletin No. 83, and it is again shown here, that slow cooling from 1100 degrees produces a higher maximum permeability and a lower hysteresis loss than cooling from slightly above 900 degrees. These facts all favor the theory that, for the kind of iron dealt with in this series of investigations, it is the removal of internal strain from the iron which is of importance in

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*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, pp. 24-27, 1915.

†"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, p. 30. Fig. 3, 1914.

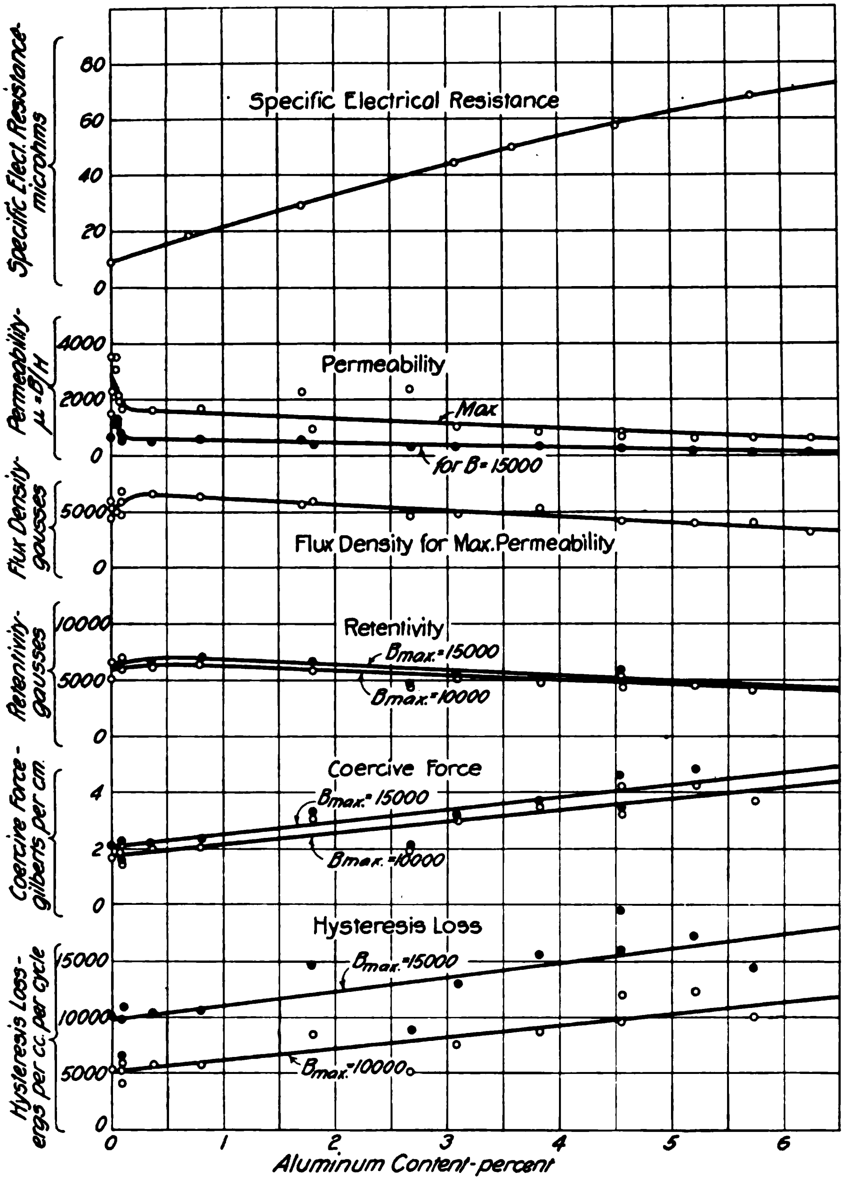

FIG. 10. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. AS FORGED

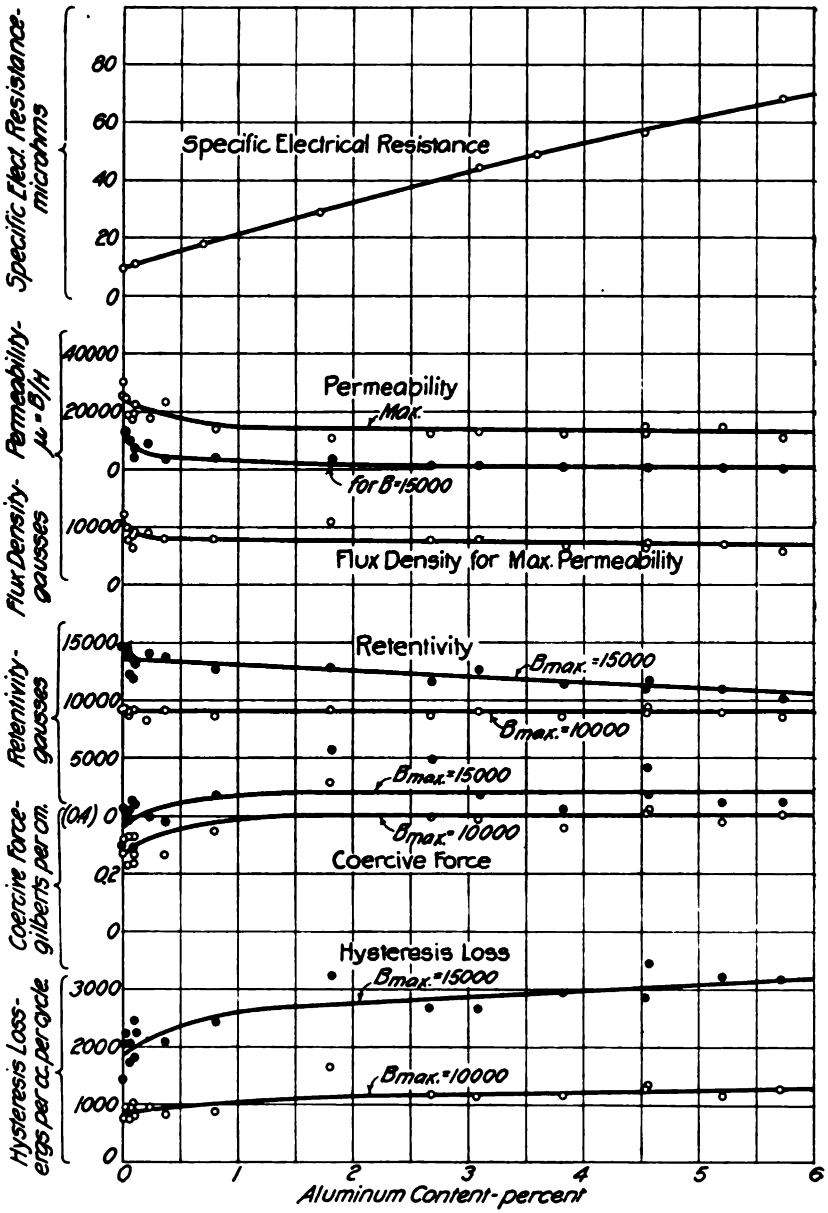

FIG. 11. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 900 DEGREES C.

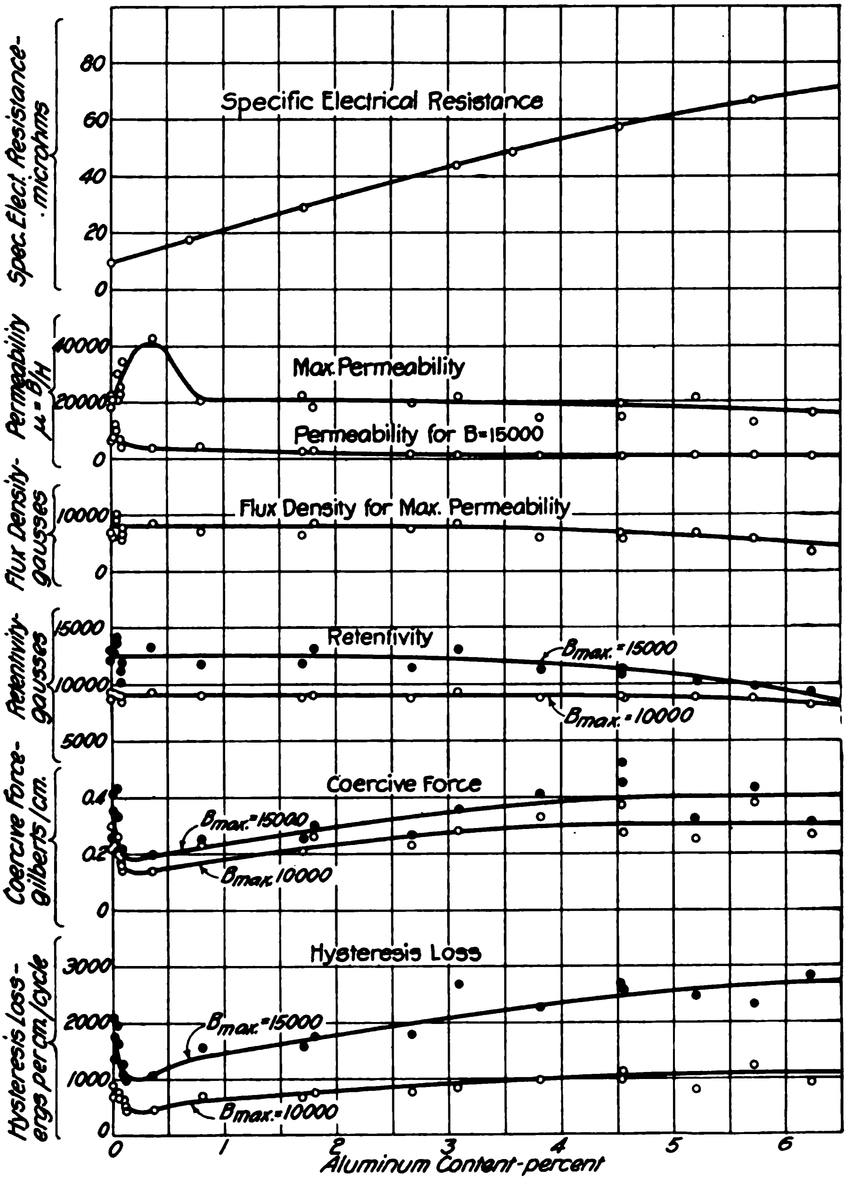

FIG. 12. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C.

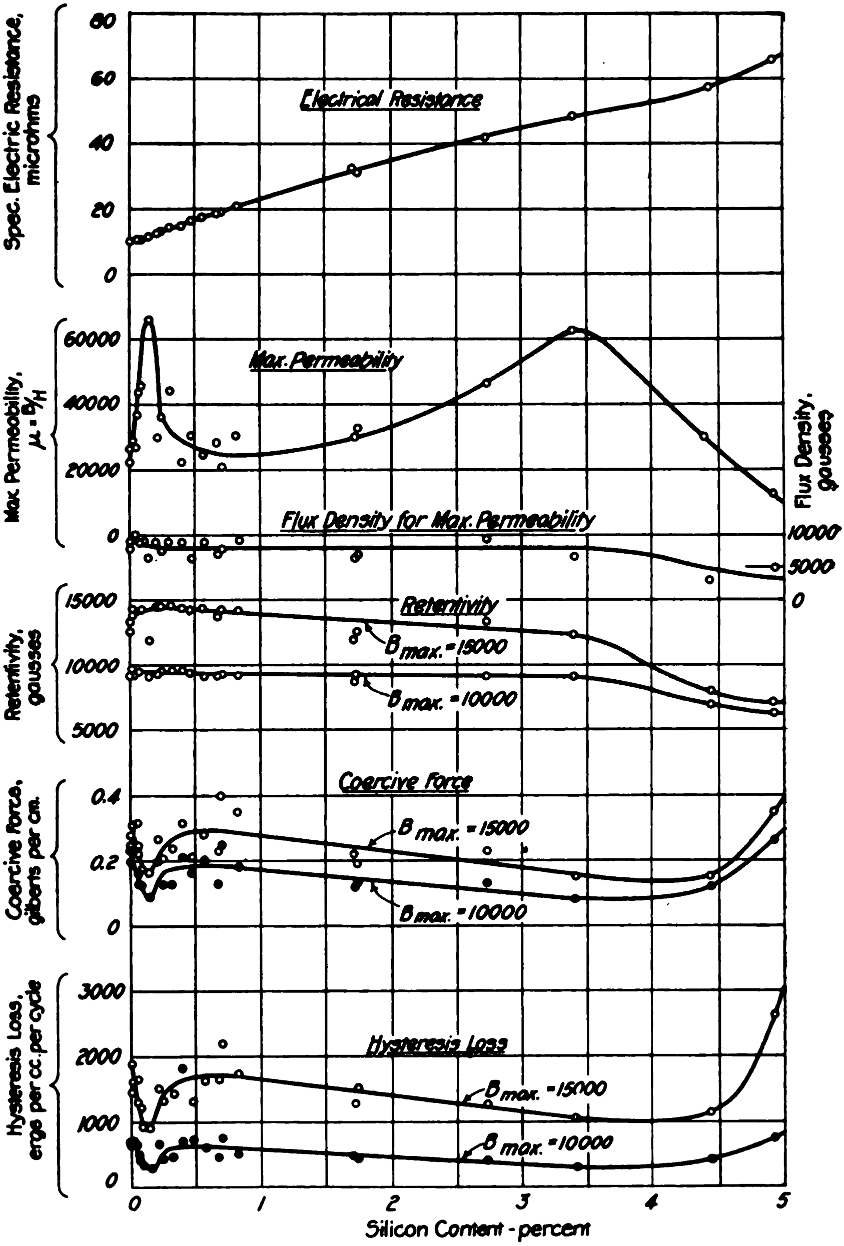

FIG. 13. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-SILICON ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C. (FROM BUL. 83)

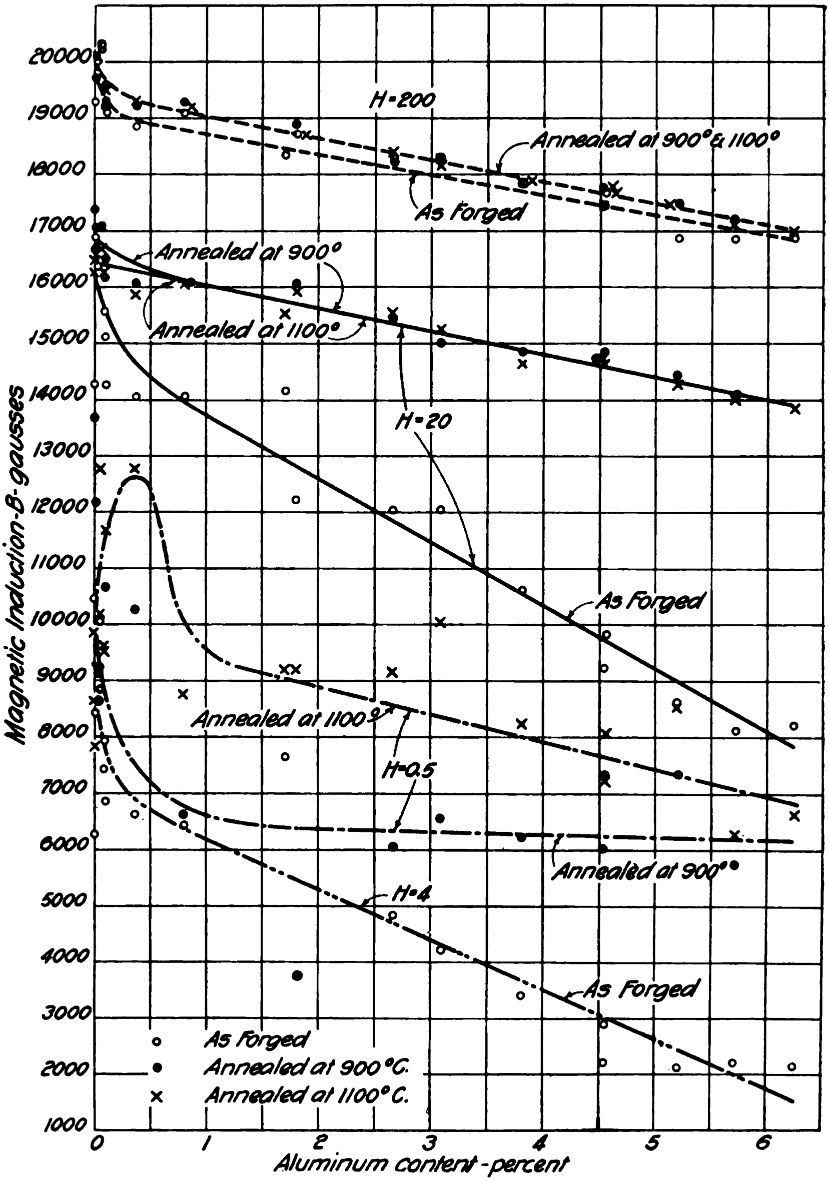

FIG. 14. FLUX DENSITY FOR VARIOUS MAGNETIZING FORCES

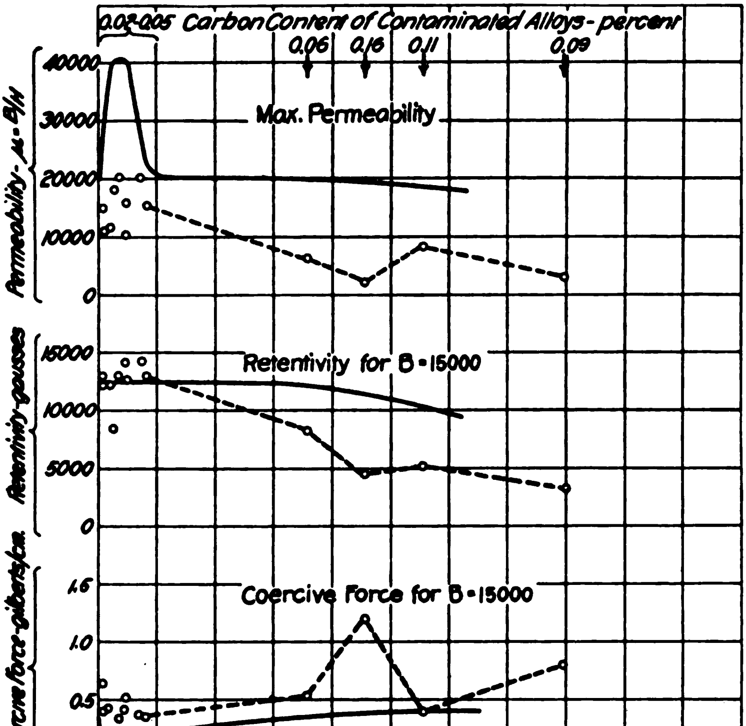

FIG. 15. MAGNETIC PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. ANNEALED AT 1100 DEGREES C.

the production of high permeability, low hysteresis iron. The transformation of all the iron into α-iron is then a foregone conclusion.

Turning now to Fig. 11, it is seen that the maximum permeability for the unannealed alloys varies from 3500 for pure iron to 600 for a 6 per cent alloy. After the alloys have been annealed at 900 degrees these values (See Fig. 12) increase to 24,000 and 13,000, respectively. The annealing at 1100 degrees (Fig. 13), although not materially affecting the pure iron, raised the maximum permeability of the 6 per cent alloy to 17,000 and that of the 0.4 per cent alloy to about 40,000. The maximum permeability is thus increased ten to thirty times, and the permeability for $B = 15,000$, two to ten times by annealing at 900 degrees. No further improvement takes place by annealing at higher temperatures: in fact, alloys low in aluminum show an actual decrease in annealing at 1100 degrees (see Fig. 14). In regard to the latter point, it has generally been found that if pure iron previously annealed at 900 degrees is annealed at a higher temperature, it decreases in permeability for values of H between 20 and 100 gilberts per cm. For $H = 200$ (Fig. 14) the permeability is the same for the 900- and 1100-degree annealing; the improvement due to annealing amounts to from 1 to 2 per cent. The saturation value is known to be practically unaffected by annealing.*

Annealing lowers the coercive force and the hysteresis loss in much the same ratio as it raises the maximum permeability, the minimum values occurring with an aluminum content of about 0.25 per cent.

When the iron-aluminum and the iron-silicon alloys (Figs. 12 and 13) are compared, the curves, as might be expected, are similar for the low alloys; both have a maximum in the maximum permeability curve and a minimum in the coercive force and hysteresis curves for about 0.2 per cent. The two sets of curves are, however, distinctly different for high alloys. The silicon curves have a second maximum or minimum for from 3.5 to 4 per cent silicon, but the aluminum curves are without such points, the maximum permeability gradually decreasing and the hysteresis loss and coercive force gradually increasing from the 0.5 per cent aluminum point.

Although the iron-aluminum series thus furnishes a high permeability, low hysteresis alloy (similar to the low silicon alloy), it furnishes none which combines these properties with a high electrical resistance in as marked a degree as does the 3.5 per cent iron-silicon alloy. Nevertheless, the 3.5 per cent iron-aluminum vacuum alloy has a maximum permeability of six times, and a hysteresis loss of only one-half, that of the commercial 3.5 per cent silicon steel. It has been shown, furthermore, that the high aluminum alloys have the advantage over the corresponding silicon alloys of being much tougher mechanically; thus they are more easily worked and lend themselves to some purposes where the silicon alloys are unsuitable on account of their comparative brittleness. The effect of carbon on the magnetic properties is illustrated in Fig. 15, where the maximum permeability, retentivity, coercive force, and hysteresis loss are shown for a number of alloys containing from 0.02 to 0.16 per cent carbon, and are compared with the uncontaminated alloys. The bars have all received the same mechanical and thermal treatment, finally being annealed at 1100 degrees. The maximum permeability attained by the alloys containing about 0.10 per cent carbon is about 5000 as compared with 20,000 for the uncontaminated alloys. The retentivity for these same alloys is much lower, and the coercive force, except in one case, much higher than for the purer alloys. The hysteresis loss is from 50 to 100 per cent and, for the low alloys containing 0.05 per cent carbon, even 200 per cent higher than for the corresponding uncontaminated alloys.

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*Gen. Elect. Rev., vol. 18, p. 881, Sept., 1915.

3. *Photomicrographs.*—Photomicrographs of a number of the alloys representative of the iron-aluminum series are shown in Figs. 16 and 17. Fig. 16 shows the uncontaminated alloys and Fig. 17 some of the contaminated ones. On the left hand side are shown the alloys as forged, and on the right the same alloys annealed at 1100 degrees C.

It is clearly seen from these figures that iron and aluminum form a solid solution as has also been shown by previous investigators. In the forged specimens the structure varies greatly, the size of the crystals generally decreasing as the aluminum content increases. The structure of 3 Al 21 is the only exception to this rule, the crystals in this case being unusually large. After the alloys have been annealed at 1100 degrees, this nonuniformity disappears and the alloys exhibit crystals of approximately the same size. For the pure iron, or for the very low alloys, 3 Al 08, 05, and 32, the structure consists, as is usual, of large crystals subdivided into many small ones.*

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*See "Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914.

"The Effect of Boron upon the Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 77, 1915.

"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

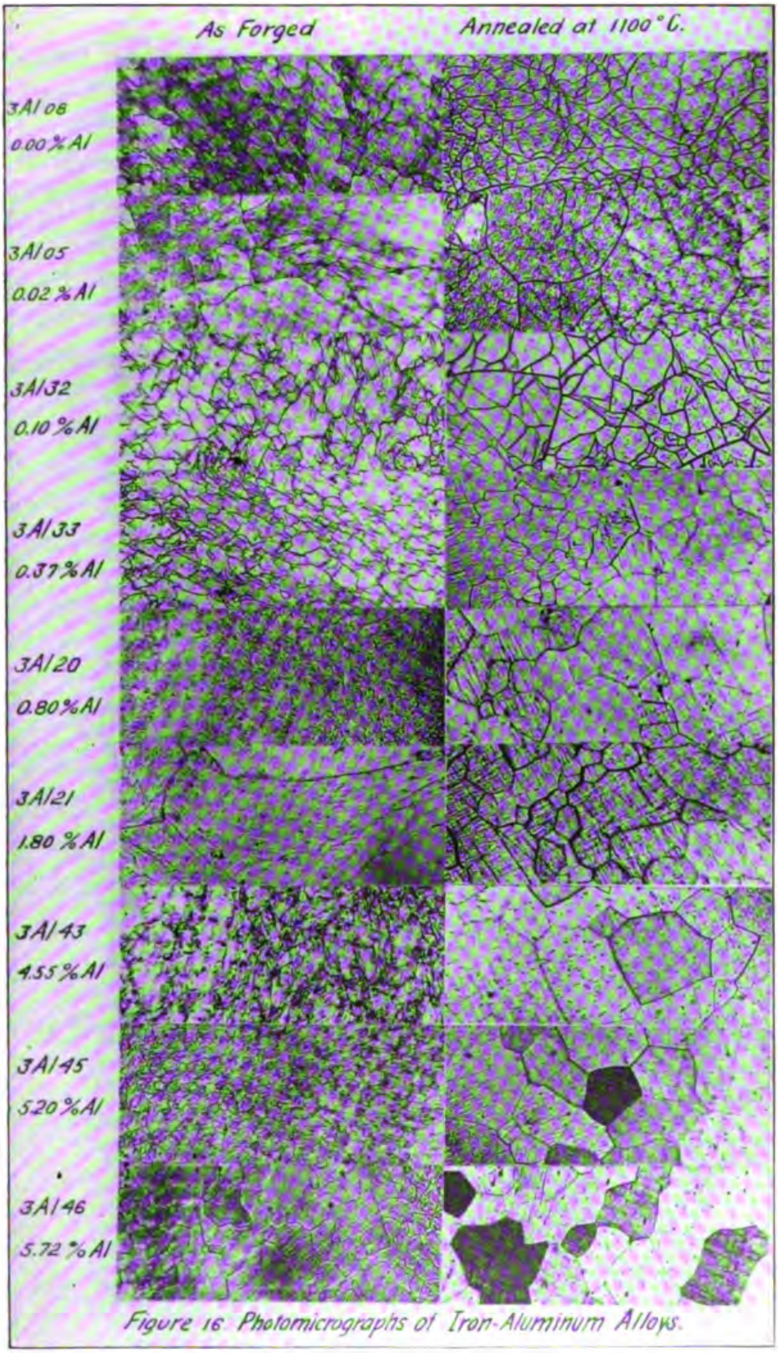

FIG. 16. PHOTOMICROGRAPHS OF IRON-ALUMINUM ALLOYS, AS FORGED AND ANNEALED AT 1100 DEGREES C. × 60

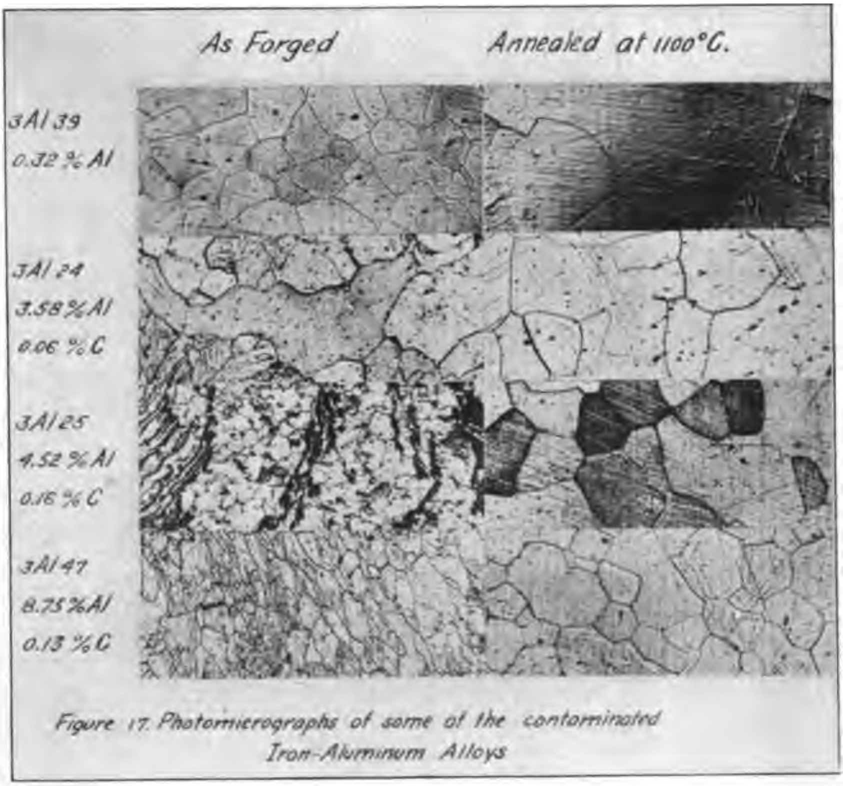

FIG. 17. PHOTOMICROGRAPHS OF SOME OF THE CONTAMINATED IRON-ALUMINUM ALLOYS, AS FORGED AND ANNEALED AT 1100 DEGREES C. × 60

For aluminum contents of 0.40 per cent and above, this subdivided structure does not appear, and the remainder of the alloys are made up of more or less regular crystals.

In the structure of the uncontaminated alloys there is no sign of impurities present; the spots that appear are evidently due to imperfect polishing. In the contaminated specimens the presence of graphite is evident only in the forged specimen of 3 Al 25, containing 0.16 per cent carbon. After annealing at 1100 degrees this evidence of graphite entirely disappeared. What becomes of the carbon in this case has not been ascertained. According to previous investigators carbon in the presence of 4.5 per cent aluminum should be completely precipitated as graphite; but, from the appearance of 3 Al 25 annealed, one is led to believe that the carbon has combined with the iron and aluminum, unless it is precipitated in such finely divided state that it is invisible under the magnification used. It is hoped that this point may be cleared by further investigation.

V. SUMMARY AND CONCLUSIONS

The results recorded in the previous pages may be summarized as follows:

a. The iron-aluminum alloys used in this investigation, prepared by the vacuum method, are less contaminated by impurities than alloys used by previous investigators. The alloys classed as uncontaminated contain only 0.01 to 0.02 per cent carbon. Other alloys containing more carbon are classed as contaminated and are used to show the effects of carbon.

b. Aluminum is more powerful as a deoxidizer than is silicon, for it does not commence to combine with iron until all oxides present are reduced. Aluminum forms a solid solution with iron throughout the range studied.

c. The tensile strength of the vacuum alloys increases in direct proportion to the aluminum content up to at least 6 per cent, the ultimate strength of the latter being 85,000 pounds per square inch (60 kg. per sq. mm.) in the unannealed state and 70,000 pounds (50 kg.) in the annealed. The corresponding figures for pure iron are 48,500 pounds (34 kg.) and 35,000 pounds (25 kg.). The toughness is only slightly affected by the aluminum content.

d. With regard to the magnetic properties aluminum, like silicon, has a very beneficial effect when added in small quantities. The best alloy obtained, containing about 0.40 per cent aluminum annealed at 1100 degrees, has a maximum permeability above 35,000. The hysteresis loss for $B_{max} = 10{,}000$ and 15,000 is 450 and 1000 ergs per cc. per cycle, respectively. For higher aluminum contents the magnetic quality decreases gradually, so that the alloy containing 3.5 per cent aluminum has a maximum permeability of 20,000 and a hysteresis loss for the given densities of 1000 and 2200 ergs, respectively. This loss is only one-half that of the 3.5 per cent commercial silicon steel.

e. The specific electrical resistance increases about twelve microhms for each per cent aluminum added. When the aluminum content exceeds 3 per cent, however, the rate of increase falls off gradually.

By the application of the vacuum process to the iron-aluminum series, an alloy, containing about 0.4 per cent aluminum, has been produced which has remarkable magnetic properties. In this respect aluminum acts like silicon. Aluminum, however, unlike silicon, yields no alloy with similar characteristics for higher percentages. On the other hand, the high aluminum alloys have the advantage over the corresponding silicon alloys of being much less brittle; and this characteristic combined with a fairly high permeability, low hysteresis loss, and an electrical resistance equal to that of the silicon alloys, may make aluminum alloys suitable for certain purposes where the silicon alloys can not be used.

APPENDIX

4. *Magnetic Testing with Burrow's Permeameter.*—During the past two years the attainment of permeabilities above 30,000 in vacuum-fused iron-silicon alloys has made the question of accuracy in results of prime importance. There are three effects which may lead to errors: (a) effect of strain, (b) end effect of the permeameter coils, (c) consequent pole effect due to nonuniformity.

a. The effect of strain has been considered in a previous bulletin.* When the rod is properly clamped, this effect is eliminated.

b. The effect upon *H* of the ends of the various permeameter coils has also been discussed in a previous bulletin.† The Bureau of Standards' correction of the Burrows method for the theoretical end effect, when the compensating current is not more than twice the current in the two main solenoids, is less than 0.1 per cent. In Bulletin No. 83 of the Engineering Experiment Station of the University of Illinois, it is shown that in some cases the current in the compensating circuit may be thirty times that in the main solenoid. In such cases the correction factor is 2.3 per cent; that is, the measured *H* must be increased by 2.3 per cent.

c. The effect of nonuniformity of the specimen is one for which no correction can be offered. It is of the same nature as the effect due to the ends of the coils, but it depends only upon the homogeneity of the rod, and, therefore, the error introduced cannot be calculated. Any nonuniformity means that lines of force will leave the iron path and complete all or part of the magnetic circuit in air. Wherever a line of force leaves or enters the iron path, a pole is developed. A line that leaves and re-enters the iron causes two poles of opposite sign. The effect of each pole at the center of the test bar varies inversely as the square of the distance from the center. Hence each pair of poles due to a leakage line has a resultant effect at the center, which depends upon the distance from the center of the points of leaving and of entering the iron. This may be illustrated by the leakage that occurs at the joint of the bar and the yoke. If the lines of force leave the rod close to the joint and enter the yoke just across the air gap, the resultant effect on the center of the bar will be small for two reasons: (*a*) the distance from the center of the resultant pole is great, and (b) the two poles tend to neutralize each other. On the other hand, if the leakage is close to the center of the bar, and the distance is great between the points of leaving the iron and re-entering it, the magnetizing or demagnetizing effect may be great. Leakage of this kind leads to a high percentage of error and is caused by the nonuniformity of the iron. A section of the iron having low reluctance is called a *soft spot;* a section having high reluctance is called a ***hard spot.***

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*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, p. 24, 1915.

†"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, p. 61, 1915.

In order to investigate the uniformity of rods, a special permeameter was constructed having movable test coils on the outside of the main solenoids. There was only one layer of winding on the solenoids and the maximum value of H obtainable without heating the coils was 60 gilberts per cm. By properly connecting the test coils it was possible to investigate the density at points along the bars and to calculate the amount of leakage at any point at any density up to that corresponding to $H = 60$.

Fig. 18 shows the results obtained with Rod No. 3 Si 27C. For each value of H the leakage along the bar was measured and plotted with the density at the center as the reference point. It should be noted that the diameter of this rod varies as much as 1 per cent, and that at densities above 15,000 the distribution of flux follows the variations in diameter. At lower densities, however, the uniformity of the material has the greater effect, because the iron is more sensitive at densities falling on the steep part of the magnetization curve. From Fig. 18 it can be seen that the assumption that compensation neutralizes only the demagnetizing effect of the leakage at the joints is entirely without justification. In this case a strong magnetizing effect is introduced because of a soft spot about four inches from the center of the rod. If the test coils marked c-c, which are wound over each end of the test bar, were placed three inches from the center they would indicate that a negative current was required for compensation. Such a current would increase the leakage at the joints and would plainly lead to error. If the coils c-c were placed five and one-half inches from the center or close to the yokes, they would then indicate that a positive current was required. However, in raising the ends of the leakage curve, the flux at the point + 4 would be raised still higher than for the uncompensated case and the effect would be to increase *H* at the center beyond its indicated value. When the two coils, c-c, are connected in series and opposed to the test coil at the center, a balance indicates that the average flux at the two ends is equal to that at the center, even though the flux at one end may be much higher than that at the other end correspondingly lower. Since when the c-c coils were placed half way between the center and the ends, a negative compensating current was sometimes required, it was decided preferable to place them nearer the ends.

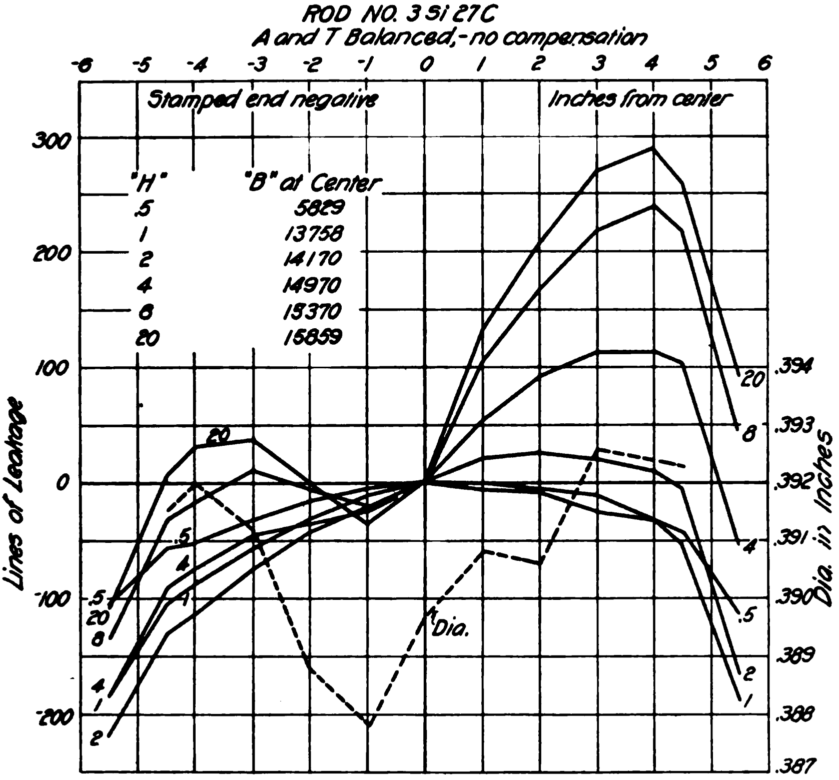

FIG. 18. CURVES SHOWING DISTRIBUTION OF FLUX. NO COMPENSATION. REFERENCE POINT AT CENTER

Fig. 19 shows the effect of compensation on the value of *H* at the

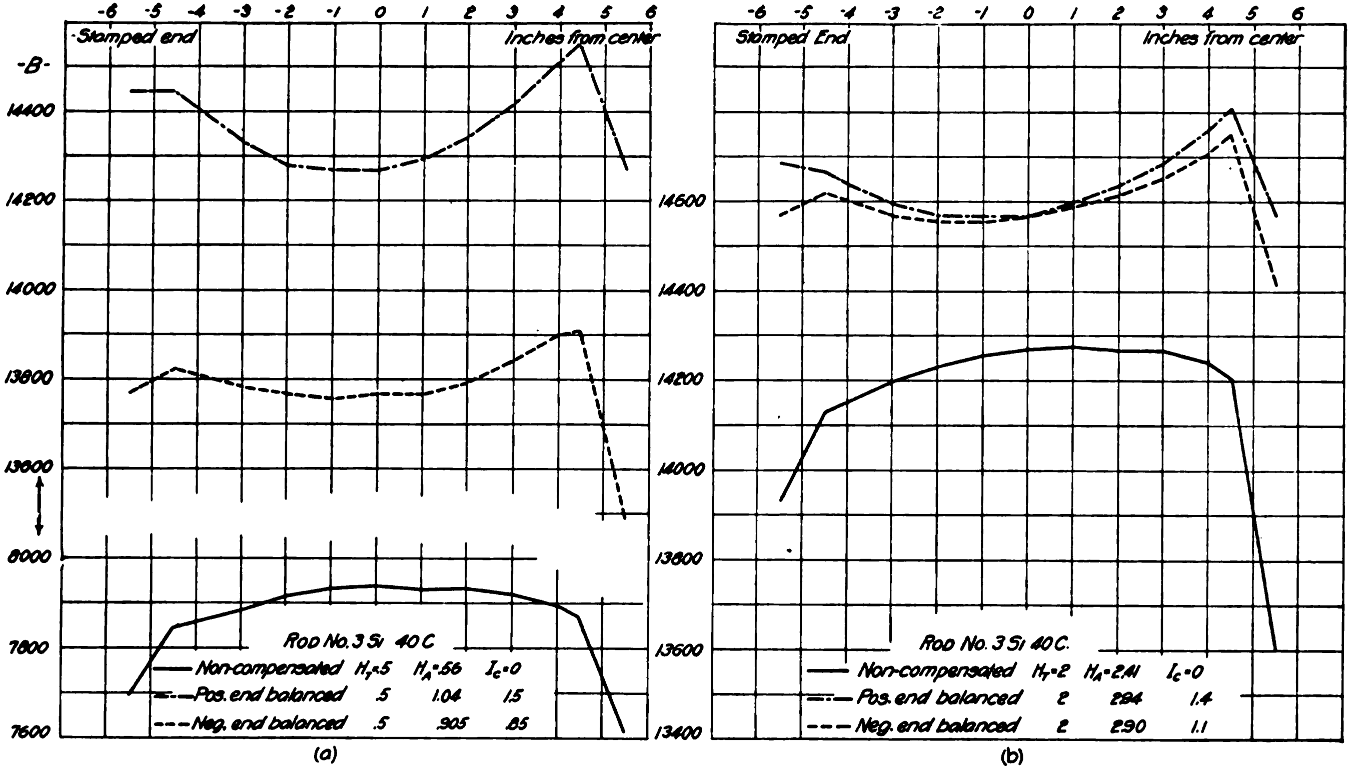

FIG. 19. CURVES SHOWING EFFECT OF COMPENSATION ON FLUX DISTRIBUTION

center for Rod 3 Si 40C. The diameter of this rod is very uniform and the nonuniform flux distribution is due to soft spots near the ends. Two conditions are shown for two values of H. In (a) H is 0.5, and in (b) H is 2.0. The test coils were connected so that the flux at either end of the test bar could be balanced against the center. The solid curve in both figures shows the distribution when the rod is uncompensated. Since the leakage is less for the negative than for the positive end, a smaller compensating current was required at the negative end in order to raise the flux to an equality with the center. The two ends were balanced separately, and the distribution for the two cases is shown by the two dashed curves. Compensation for the positive and negative ends raised the density at the center from 7,870 to 14,280 and 13,680, respectively, for $H = 0.5$; and to 14,280 and 14,580, respectively, for $H = 2.0$. An increase was desired after the leakage at the joints had been neutralized, but there was no way of knowing how much magnetizing effect the "soft spot" had on the center. The density at the center is probably exaggerated, especially for $H = 0.5$.

The double bar and yoke method is based upon the assumption that the rods are uniform. If the rods are uniform, when the c-c coils are placed about half way between the center and the ends, a balance of the average of the flux between these points and the center will result. There may still be some leakage near the joints, but for reasons stated previously its effect is small. As shown in Fig. 18, the distribution can not be assumed to be uniform, and so results by this method are likely to be inconsistent. Nonuniformity is most troublesome in rods of high permeability; with rods of ordinary quality this effect is not so important. For commercial work where only commercial iron is tested, this method is probably sufficiently accurate; but for laboratory testing of high permeability iron, the limiting accuracy should be carefully considered.

5. *Results Obtained with Rings.*—In Table 8 will be found results obtained with rings made of pure open hearth iron, remelted in vacuo. For the sake of comparison, the results obtained with some of the same iron in the form of rods tested by the Burrows method have been included. All the specimens have been annealed at 1100 degrees. C. From these results it is seen that for permeabilities not exceeding 20,000, when the possible variation in

mechanical and heat treatment of the various specimens are considered, the two methods check very well.

TABLE 8

COMPARISON BETWEEN THE RING METHOD AND BURROW'S METHOD OF MAGNETIC TESTING

Specimen No.	Kind of Specimen	Max. Permeability	Density for Max. Permeability	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gausses		Coercive Force Gilberts per cm.		Remarks
				For B = 10000	For B = 15000	For B = 10000	For B = 15000	For B = 10000	For B = 15000	For B = 10000	For B = 15000	
Open Hearth Iron Remelted in Vacuo and Annealed at 1100 Degrees C.												
4-01	Ring	14300	8500	13700	5700	986	2063	8400	12300	0.33	0.39	
4-01	Rod	14180	8500	13200	5350	1080	2190	8700	12300	0.37	0.40	
4-02	Ring	16500	9500	16450	6400	935	2010	8700	13900	0.30	0.35	
4-03	Ring	17000	9000	16700	8250	852	1755	8400	12600	0.28	0.33	
4-03	Rod	20900	9000	20200	7500	865	1760	9300	13600	0.30	0.34	
4-04	Ring	16300	10000	16300	6000	870	1880	8400	13300	0.30	0.35	
Iron-Aluminum Alloys Melted in Vacuo and Annealed at 1100 Degrees C.												
3 Al 22	Ring	20500	7000	16700	900	636	1320	8400	11000	0.19	0.20	Al-0.45%
3 Al 18	Ring	19700	6400	12500	350	674	1210	9000	10200	0.20	0.20	Al-0.23%

6. *Determination of Aluminum in Iron, by J. M. Lindgren.*—Four well known methods for the determination of aluminum in iron were tried. First, precipitation by means of sodium hydroxide; second, fusion in a nickel crucible with sodium hydroxide; third, the electrolytic separation with a mercury cathode; and finally, the method of Rothe,* with ether as a solvent for ferric chloride. The first two methods were objectionable because of the slow, tedious filtrations and the resulting voluminous filtrates. The electrolytic method did not remove all the iron, and so made a determination of that constituent necessary. Here, also, tedious filtrations and voluminous filtrates were encountered. Gooch and Hovens† made use of ether in the determination of aluminum by precipitating $AlCl_36H_2O$ with gaseous hydrochloric acid and completing the separation from iron by means of ether.

Rothe demonstrated that by means of ether large amounts of iron could be separated from nickel-aluminum, copper-cobalt, vanadium, and titanium. He employed a special apparatus for the separation, which consisted of two separatory funnels joined to a common stem with a 3-way stopcock. By means of pressure applied by the vapors of ether the solution was forced into the second funnel, where the separation was continued by means of fresh ether. Rothe demonstrated that the iron should be present in the ferric state and that the hydrochloric acid present should have a definite strength.

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*Chem. News, vol. 66, p. 182, 1893.

†Chem. News, vol. 74, p. 296, 1896.

In the present work the separation was carried out in a 200 cc. Jena beaker. It was found necessary, in transferring from one separatory funnel to another during the process of removing all the iron, to use considerable amounts of hydrochloric acid in order to wash the syrupy aluminum chloride from the sides of the funnel; this increased the acid concentration, and made it very difficult to remove all the iron. It occurred to the writer that the separation could be made easier in the original beaker in which solution was effected by decanting the supernatant ether and ferric chloride, and so make possible a cleaner and easier separation by subsequent washings with ether in the same beaker. The decantation was more easy to accomplish than first appeared possible, because the supernatant liquid of ferric chloride and ether was much lighter than the lower solution of aluminum chloride and acid.

TABLE 9

DETERMINATION OF ALUMINUM IN IRON

Per Cent Aluminum Taken	Per Cent Aluminum Found
.732	.763
.732	.791
1.464	1.484

Details of the method used are as follows: Dissolve from 1 to 5 grams of iron in a 200 cc. Jena beaker by means of hydrochloric acid, with additions of nitric acid to afford complete oxidation of the iron; evaporate to dryness on a water bath, and take up with just enough hydrochloric acid, sp. gr. 1.19, to completely dissolve the iron. Care should be taken that no iron oxide remain undissolved. Evaporate again on a water bath to a syrup consistency, add 30 cc. of ether, and stir vigorously with a glass rod until the ether has dissolved as much ferric chloride as possible. Allow to stand until the separation is distinct and both layers are clear. At this point it is sometimes necessary to add more hydrochloric acid, two cc. at a time, until the solutions become perfectly clear. Carefully decant the supernatant solution of iron and ether and repeat the addition of ether. Usually six additions of ether are sufficient to extract completely all of the iron. Evaporate the ether and precipitate the aluminum in the usual way by means of ammonia. Burn the oxide in a porcelain crucible and finally in a muffle at 1000 degrees C to constant weight. The necessity of a high temperature in burning the aluminum precipitate shows the impossibility of obtaining satisfactory results by any of the methods involving the burning off of a mixed precipitate of aluminum and iron.

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THE EFFECT OF MOUTHPIECES ON THE FLOW OF WATER THROUGH A SUBMERGED SHORT PIPE

BY
FRED B. SEELY

CHARTERED 1867

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ENGINEERING EXPERIMENT STATION

BULLETIN No. 96 APRIL, 1917

THE EFFECT OF MOUTHPIECES ON THE FLOW OF WATER THROUGH A SUBMERGED SHORT PIPE

BY
FRED B. SEELY
ASSOCIATE IN THEORETICAL AND APPLIED MECHANICS

ENGINEERING EXPERIMENT STATION
PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

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THE EFFECT OF MOUTHPIECES ON THE FLOW OF WATER THROUGH A SUBMERGED SHORT PIPE

I. INTRODUCTION

1. *Preliminary.*—This bulletin presents the results of experiments on the flow of water through a submerged short pipe with and without entrance and discharge mouthpieces of a variety of angles and lengths. It treats of the loss of head which occurs when a stream contracts or expands under differing conditions of flow and emphasizes the marked effect that turbulence of flow may have upon the amount of head lost. The discussions have a direct bearing upon various problems in hydraulic practice which involve the contraction and expansion of a stream in flowing through passages.

Comparatively little experimental work has been done to determine the value of conical mouthpieces of various angles and lengths in reducing the lost head at the entrance to and discharge from a submerged pipe, particularly for mouthpieces of the sizes and proportions comparable with those met in engineering practice. The need for such experiments is, therefore, apparent. The minimizing of the lost head due to the contraction and expansion of a stream may be of considerable importance in a variety of hydraulic problems; for example, the intake to a pipe particularly when the pipe is of short length and of large diameter, the suction and discharge pipes of a low head pump, the reduction or expansion from one pipe to another of different diameter or of different shape, the passages through a large valve, the passages through locomotive water columns, the draft tube to a turbine, the connection from a centrifugal pump to a main, the sluice ways through dams, the slat screens at head gates, culverts and short tunnels, jet pumps, the Boyden diffuser as formerly used for the outward flow turbine, the Venturi meter, the suction and discharge pipes of dredges, and the guide vanes and runner of a turbine.

Losses due to this cause are difficult to estimate and easy to overlook. Even where such losses are in themselves of little consequence as compared with other quantities involved, they may have a considerable influence upon subsequent losses on account of the turbulent motion started by the contraction or expansion. The efficiency of a drainage pump or other low head pump, for example, may be increased by an entrance mouthpiece on the suction pipe because it allows the pump to receive the water in a smoother condition of flow. It is well known that a turbine must receive the water from the guide vanes without shock if, in the subsequent flow through the runner, the energy of the water is to be absorbed efficiently by the turbine. The loss of head through a Venturi meter may be considerably increased if the meter is placed too short a distance downstream from a valve, elbow, or other obstruction or cause of disturbance in the pipe. The friction factor for a pipe following an obstruction or bend may be changed by the disturbance thus caused; the lost head at the entrance to a pipe, particularly when inward-projecting, may be more than that ordinarily assumed for a tube three diameters long. There is but little definite knowledge on the whole subject of the effect of abnormal conditions, and it offers a large scope for investigation. The fact that a comparatively small change in the form of the blades of a turbine runner may result in a large effect on the efficiency of the turbine should prove suggestive when estimating the probable effect of turbulent flow in less severe or critical cases. It is also worth mentioning in this connection that the recent advances in turbine design have been due largely to the attention given to the approach channels to the guide vanes and to the design of the draft tube.

The flow of water usual in engineering practice is more or less turbulent. The general equation of energy, or Bernoulli's theorem, so generally used in hydraulics, applies only when the particles of water move with uniform velocity in parallel stream-lines. Although this condition of flow seldom occurs, satisfactory analyses may often be made by using an average velocity and introducing empirical constants. However, a very slight change in the conditions under which flow takes place may cause, in some cases, a large difference in the action or behavior of the water. There is, therefore, always danger in extending the use of experimental data or empirical constants to apply to conditions of flow quite different from those under which the data were obtained.

2. *Acknowledgment.*—All the experimenting was done in the hydraulic laboratory of the University of Illinois under the general direction of Professor A. N. Talbot. A part of the problem and some

of the methods had been developed by Professor Talbot through experimental work which had been carried on in the hydraulic laboratory for a number of years. The types and proportions of the mouthpieces and the general features of the apparatus had been planned by him. Thesis work of the following students has been utilized as preliminary material in helping to determine the methods used in the investigation: W. P. Ireland, "Entrance Head in Pipes and Conduits," 1903; C. C. Wiley, "Entrance Head and Discharge Head in Pipes," 1904; W. R. Robinson, "Entrance Head and Discharge Head in Pipes," 1906; W. R. Robinson, "An Investigation of the Flow of Water through Submerged Orifices and Pipes," 1909. Although but few of the data given in these theses were incorporated in the final results as reported in this bulletin, they were of considerable value in determining the influence of certain factors involved in the method of experimenting.

The major part of the experimenting was carried out by the writer during 1914 and 1915, with the help of Mr. L. J. Larsen and Mr. R. L. Templin, research fellows in the Engineering Experiment Station, whose careful work is gratefully acknowledged.

II. APPARATUS AND METHOD OF EXPERIMENTING

3. *Short Pipe.*—The cast-iron short pipe to which the mouthpieces were attached was 22½ in. long, bored to a smooth surface and to a 6-in. diameter; an average of thirty micrometer readings taken across three diameters at each of ten successive sections along the pipe gave 0.5995 in. It was threaded at each end so that a mouthpiece could be screwed on either end or on both ends. Fig. 1 shows the 6-in. short pipe used in the experiments. Fig. 5 is from a photograph and shows the tank used in the experiments and also the 6-in. short pipe and some of the mouthpieces. A flange near the middle of the pipe was used to attach it to a partition separating the two compartments of the tank, allowing the pipe to project into each compartment. Fig. 2 shows the short pipe in place in the tank with mouthpieces attached.

Experiments were also made on a steel tube 3.11 in. in diameter by 12 in. long used as an inward-projecting short pipe only with no mouthpiece attached.

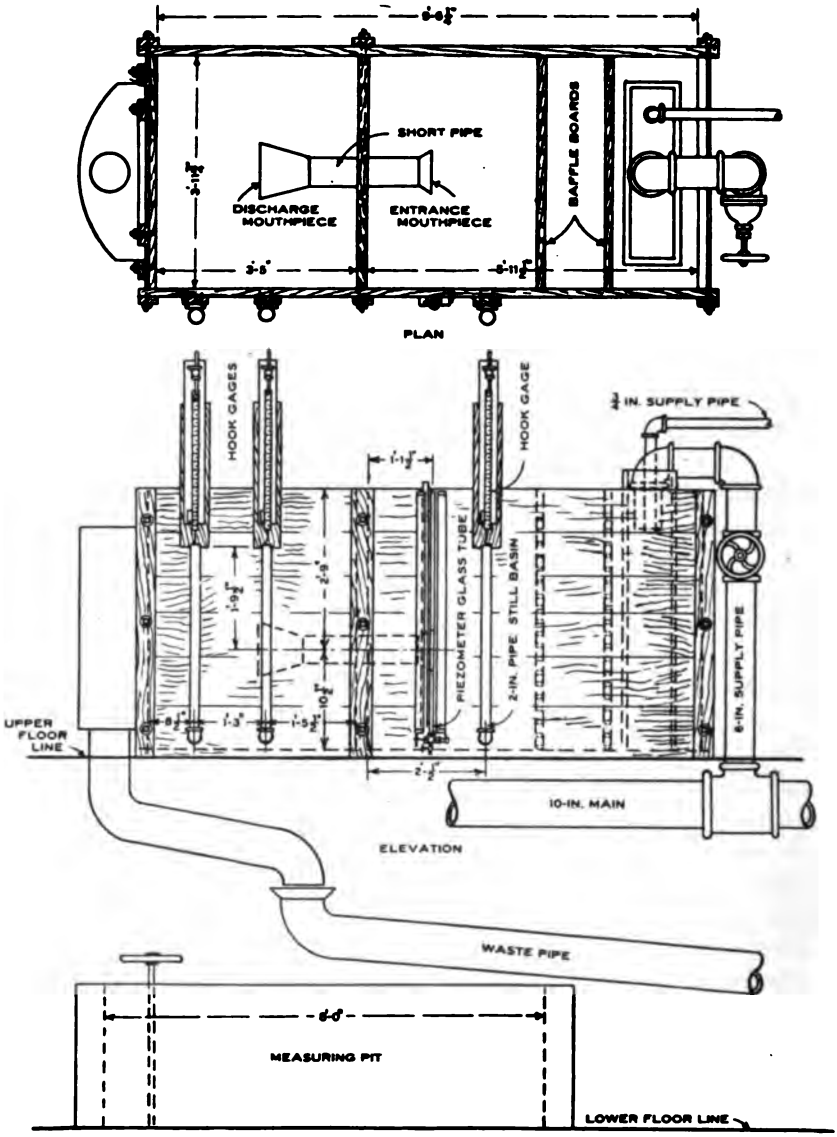


FIG. 2. TANK AND ARRANGEMENT OF APPARATUS

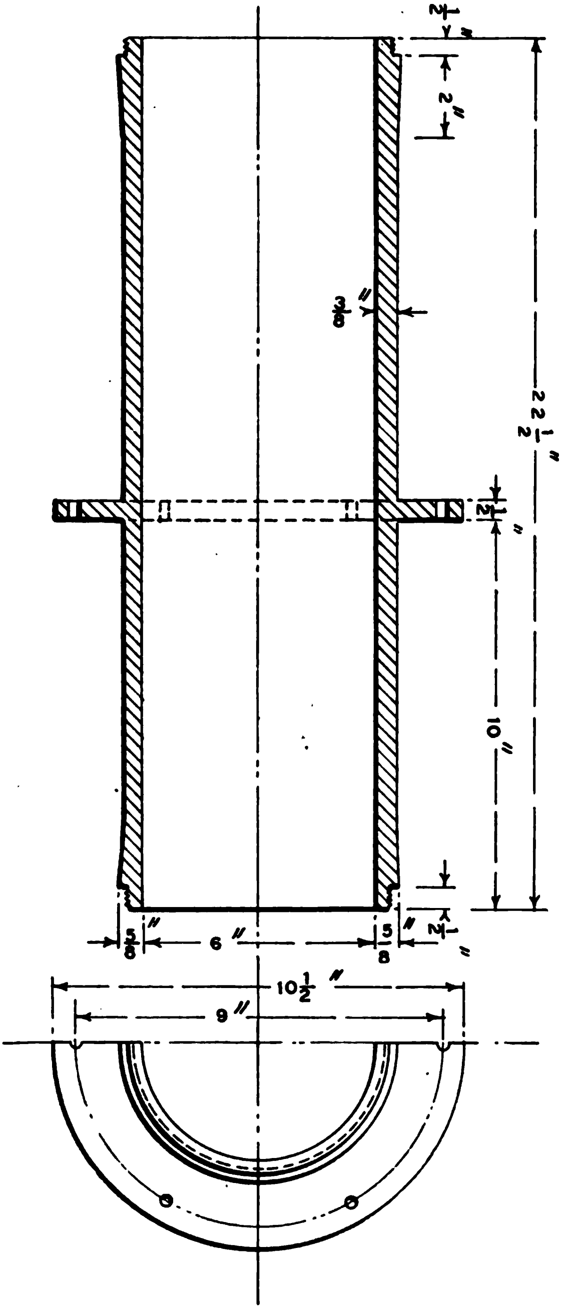
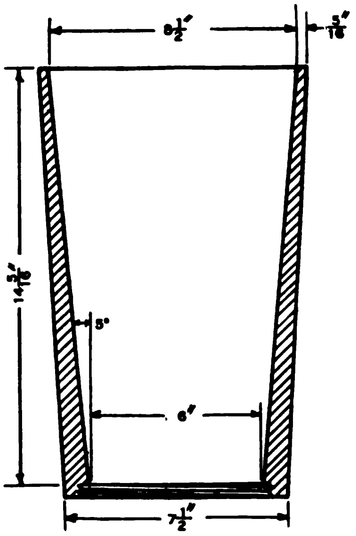


Fig. 1. Short Pipe to which Mouthpieces Were Attached



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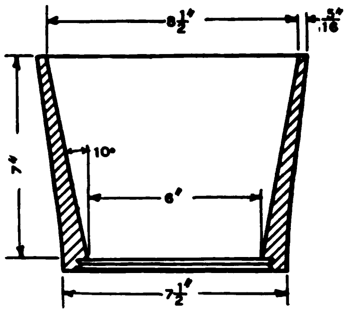

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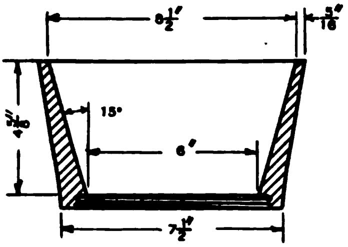

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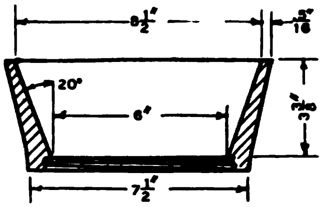
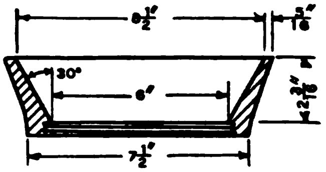
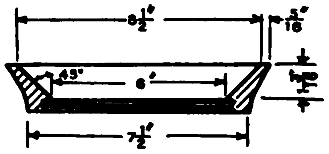


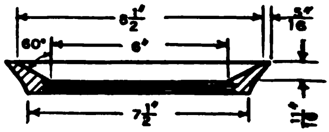
20° MOUTHPIECE



30° MOUTHPIECE

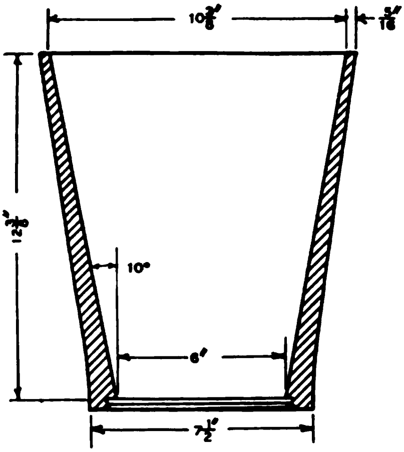


45° MOUTHPIECE

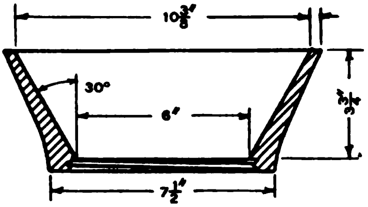


60° MOUTHPIECE

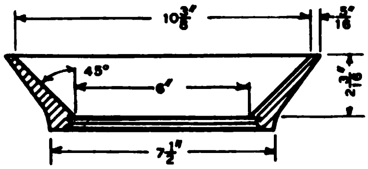
FIG. 3. CONICAL MOUTHPIECES HAVING AN AREA RATIO OF 1 TO 2



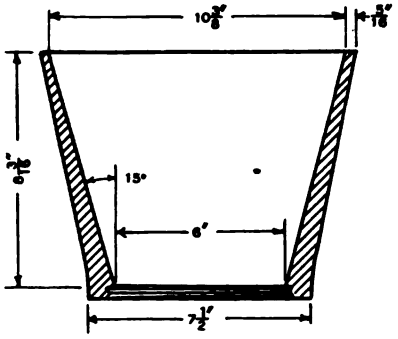
10° MOUTHPIECE



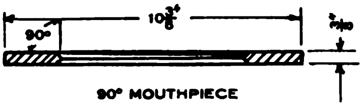
30° MOUTHPIECE



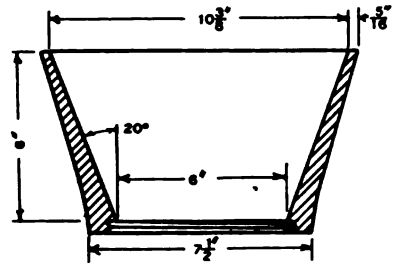
45° MOUTHPIECE



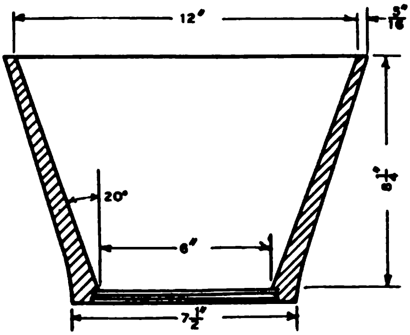
15° MOUTHPIECE



90° MOUTHPIECE



20° MOUTHPIECE



20° MOUTHPIECE
[Area Ratio 1 to 4]

FIG. 4. CONICAL MOUTHPIECES HAVING AN AREA RATIO OF 1 TO 3

4. *Mouthpieces.*—The cast-iron conical mouthpieces which were screwed on the ends of the 6-in. pipe are shown in Figs. 3 and 4. The smallest cross-sectional area of each mouthpiece was the same as the area of the pipe with which the mouthpiece made a smooth connection. The largest or outer area of one series of mouthpieces was twice the area of the pipe, that is, the ratio of the area of the pipe to the largest area of the mouthpiece was 1 to 2. Another series had an area ratio of 1 to 3, and one mouthpiece (20 degree angle) had an area ratio of 1 to 4. The length of a mouthpiece for any area ratio depends, of course, upon the angle of the mouthpiece. Table 1 gives a list of the mouthpieces used, together with the particular combinations of an entrance and a discharge (exit or diverging) mouthpiece employed. In no case was a mouthpiece used alone, that is, without being attached to the short pipe. It will be noted that any mouthpiece could be attached to the entrance end of the short pipe only, or to the discharge end only, or two could be attached, one on each end of the pipe. In the case of a few of the mouthpieces, duplicates were made. The angle of the mouthpiece as given in Table 1 and in the various figures, and as used in these pages, means the angle between the axis line of the pipe and one element of the cone, not the total angle of convergence or divergence. Hence, a 90-degree entrance mouthpiece is a flat disc giving a square or flush entrance to the pipe.

TABLE 1

LIST OF MOUTHPIECES USED ON THE SHORT PIPE

Each Mouthpiece Tested Singly on Entrance End and Discharge End				Combinations			
				Entrance End		Discharge End	
Area Ratio	Angle of Mouthpiece Degrees	Area Ratio	Angle of Mouthpiece Degrees	Degrees	Area Ratio	Degrees	Area Ratio
1 to 2	5	1 to 3		20	1 to 2	5	1 to 2
	10		10	20	1 to 3	5	1 to 2
	15		15	20	1 to 2	10	1 to 3
	20		20	20	1 to 2	10	1 to 2
	30		30	30	1 to 2	10	1 to 2
	45		45	20	1 to 2	15	1 to 3
	60			15	1 to 2	15	1 to 2
			90	20	1 to 2	20	1 to 4
				20	1 to 2	20	1 to 3
		1 to 4	20	20	1 to 3	20	1 to 2
				30	1 to 2	30	1 to 2

5. *Tank and Method of Experimenting.*—The same tank was used in all the experiments. The dimensions of the tank are shown in Fig. 2 and a photograph gives other details in Fig. 5. The tank was divided into two compartments by a vertical partition to which the short pipe was attached in a horizontal position.

Water from the laboratory standpipe was supplied to the tank through a 6-in. supply pipe and also through a ¾-in. pipe, the latter making possible a finer adjustment in maintaining a constant head. After passing through baffle boards the water flowed through the short pipe and finally left the downstream compartment by passing out of the small openings in the end of the tank, the flow through which was regulated by placing stoppers in some of the holes. These holes were arranged in two long vertical rows in the end of the tank, one near each side, and the stoppers were arranged so as to give nearly the same distribution of flow from each row. This, it was found, helped to maintain steady conditions.

The quantity of water discharged was measured in a pit about 6 ft. deep, with a diameter of 7.995 ft. as obtained from readings of a micrometer attached to a rigid stick. The rise in the pit was determined by a vertical graduated rod which was read directly to 0.02 ft. and to 0.004 ft. by estimating. A float was attached to the bottom of the rod and a still basin was provided. The water was wasted into another pit through a movable spout until the surface of the water in the measuring pit became fairly still so that an accurate reading of the rod could be taken. A hook gauge was used to test the accuracy of the float and rod. At the end of the experiment the water was again wasted in the same manner. A calibrated stop watch gave the time corresponding to the rise in the pit.

The head causing flow through the short pipe is the difference in the levels of the water surfaces in the two compartments of the tank. The head was measured in nearly all the experiments by means of hook gauges. These gauges were read directly to 0.001 ft. and to 0.0005 ft. by estimating. Vertical 2-in. pipes attached toward the bottom of the tank served as still basins for the hook gauges (see Figs. 2 and 5). The level of the water in the upstream compartment was determined by the use of one hook gauge only, but two gauges were used on the downstream compartment in the earlier experiments. It was found, however, that for the lower heads the two gauges gave practically the same result and for the higher heads the gauge nearer the partition gave less fluctuation. For these reasons, and because of less difficulty in getting simultaneous readings of only two gauges, it was decided to take readings with one gauge only on each compartment.

Zero readings of the hook gauge were obtained by reading the gauges when the tank was nearly full and when no water was allowed to escape, the levels of the water surfaces in the two compartments then being the same. Zero readings were taken frequently during the experiments.

For most of the heads above 0.3 ft. the head was measured by a differential gauge which was connected to each compartment of the tank by means of rubber hose. A mixture of carbon tetrachloride and gasoline having a specific gravity of 1.25 was used; thus making the gauge reading four times the head. The gauge was provided with a scale graduated to 0.005 ft. In a few cases two vertical piezometer glasses were used, one attached near the bottom of each compartment, the difference in readings of which (corrected for zero reading) gave the head to 0.001 ft. These three methods overlapped somewhat so that certain heads were measured by all three methods.

Leakage from the tank and from the measuring pit was determined several times during the progress of the experiments and was found to be negligible.

The following procedure comprised an experiment: Stoppers were removed from the end of the tank in sufficient number to give the desired discharge, and the inflow through the 6-in. and ¾-in. pipes was then adjusted until the difference in levels of the water surfaces in the two compartments of the tank became constant. The ¾-in. supply pipe was used to make the final adjustment of the head and to hold the head constant throughout the experiment. After obtaining a constant head, the waste pipe shown in Fig. 2 was pulled from beneath the discharge pipe, allowing the water to discharge into the measuring pit until the rise in the pit was sufficient to allow of its measurement without appreciable error, and to allow an accurate measurement of the head to be made. The head was taken as an average of from two to ten readings of the hook gauges, the larger number being necessary with the higher velocities on account of the greater fluctuations of the water levels due to the more turbulent conditions of the water, especially in the downstream compartment. Each experiment was repeated three times, as a rule, although in the case of some of the small-angle discharge mouthpieces and at the higher velocities, as many as six or eight runs were made.

III. Experimental Results and Discussion

6. *Brief Analysis of Flow.*—The term short pipe or tube as technically used in hydraulics applies to a tube merely long enough to allow the stream to expand after contraction at entrance and flow full at the discharge end. A length equal to three diameters is usually considered sufficient although the pipe used in these experiments was 3.75 diameters long.

The head, *h*, causing flow through a tube is divided into two items, (1) the velocity head in the tube, which represents energy per pound of water transformed from potential or pressure energy into kinetic energy, and (2) lost head,* *h′*, representing energy per pound of water dissipated (transferred into heat) chiefly by impact and friction of the water particles. The velocity head is usually represented by $\frac{v^2}{2g}$, where v is the mean axial velocity in the pipe. It is clear that if the water has a mean axial velocity, v, and at the same time is in a turbulent state of motion, probably also with some rotation as a whole, the water will possess more kinetic energy than if parallel stream-line flow occurred with the same axial velocity. Hence with turbulent flow, $\frac{v^2}{2g}$ does not represent the total kinetic energy created. Furthermore, since dissipation of energy (lost head) accompanies the transfer of energy, it is evident that more head will be lost when turbulent flow is produced and also wherever the velocity head is transferred back into pressure head as in reducing the velocity by means of a diverging mouthpiece. It is plain that the flow of water through a short pipe with or without mouthpieces does not yield to a simple detailed analysis.

In equation form we have, then,

$$h = h' + \frac{2g}{v^2} \quad \ldots\ldots\ldots \quad (1)$$

$$\text{But } v = c\sqrt{2gh}. \quad \ldots\ldots\ldots \quad (2)$$

---

*The term "lost head" is sometimes used to denote the head which causes the flow, but as used in these pages it will always mean dissipated energy.

$$\text{Therefore, } h' = \left(\frac{1}{c^2} - 1\right)\frac{v^2}{2g}. \quad . \quad . \quad . \quad . \quad (3)$$

in which c is the coefficient of discharge since the pipe flows full at the discharge end, and is the ratio of the measured rate of discharge, q, to the theoretical rate of discharge, or, in equation form,

$$c = \frac{q}{a\sqrt{2\ gh}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

in which a is the area of the cross-section of the pipe. The expression $\left(\frac{1}{c^2} - 1\right)$ is called the coefficient of loss and is denoted by m.

If a converging or entrance mouthpiece is attached to the short pipe, the contraction of the stream will be somewhat suppressed. The discharge, therefore, will be increased, with a corresponding reduction in the lost head. How much of this decrease in energy loss occurs at the entrance and how much during the subsequent flow in the pipe for any given case is difficult to say. The loss of head which would occur in a 6-in. pipe 22½ in. long, if it were a part of a longer pipe and preceded by a considerable length of straight pipe of the same diameter, would be that due to pipe friction,

$$f\frac{l}{d}\,\frac{v^2}{2g} \text{ or } 0.086\frac{v^2}{2g}.$$

This loss is not considered in these experiments for, although the pipe may flow full for the greater part of its length when the contraction at entrance is largely suppressed by an entrance mouthpiece, the state of flow is no doubt determined chiefly by the entrance conditions and the loss of head should all be considered as entrance loss.

If a diverging mouthpiece is attached to the discharge end of the short pipe, a part of the velocity head in the pipe may be regained. Theoretically, the amount possible of recovery is the difference between the velocity head in the pipe, $\frac{v^2}{2g}$, and the velocity head at the outer or discharge area of the mouthpiece, $\frac{v_0{}^2}{2g}$. This may be

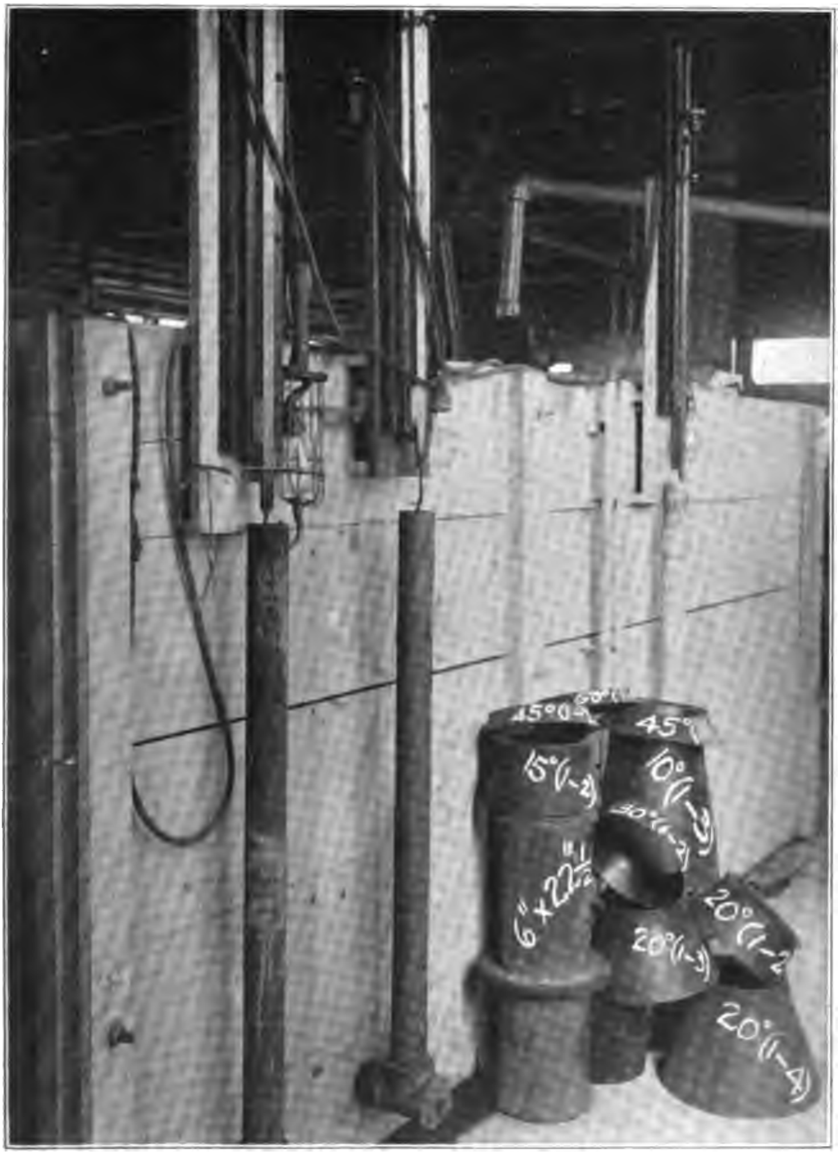


FIG. 5. VIEW OF TANK, OF SHORT PIPE, AND OF SOME OF THE MOUTHPIECES

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expressed as

$$\left[1-\left(\frac{a}{A}\right)^2\right]\frac{v^2}{2g},$$

in which $\frac{a}{A}$ is the ratio of the area of the pipe to the outer area of the discharge mouthpiece. In other words, if the water approaches the mouthpiece with parallel stream-lines (smooth flow) and if the mouthpiece could reduce the velocity without eddying—ideal conditions—a discharge mouthpiece with an area ratio of 1 to 2 would regain 75 per cent of the velocity head in the pipe, while a mouthpiece having an area ratio of 1 to 3 would regain 88 per cent of the velocity head and a 1 to 4 mouthpiece would recover 94 per cent.

The flow in a short pipe, particularly when no mouthpiece is attached to the entrance end, is far from smooth. And while this turbulence of flow would be expected to change materially the action or effect of the discharge mouthpiece from that occurring under ideal conditions, there is no rational method of taking this into account. It seems probable, however, that the discharge mouthpiece influences the flow for some distance back in the pipe so that the loss of head in the tubes is less than when no discharge mouthpiece is used. This action in turn allows the mouthpiece to act with greater efficiency. This will be discussed later.

It should be noted that in the expression for the coefficient of loss,

$$m=\left(\frac{1}{c^2}-1\right). \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

c is the coefficient of discharge based on the area of exit from the system. Hence in the case of the short pipe with a discharge mouthpiece attached, the lost head is expressed in terms of the velocity head at exit from the mouthpiece. But since $av = Av_0$ we have the lost head expressed in terms of the velocity head in the pipe as

$$h'=\left(\frac{1}{c_0^2}-1\right)\left(\frac{a}{A}\right)\frac{v^2}{2g} \quad . \quad . \quad . \quad . \quad . \quad . \quad (6)$$

7. *Tables and Figures.*—Table 2 gives the condensed experi-

TABLE 2

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING CONICAL MOUTHPIECES

(Depth of Submergence, about 3½ Diameters)

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	h	q	v	c
None (Inward-Projecting)	None	.0074	.102	.520	.756
		.0128	.135	.689	.757
		.0153	.153	.778	.779
		.0190	.170	.868	.785
		.0292	.210	1.07	.785
		.0327	.223	1.14	.784
		.0563	.294	1.50	.785
		.0745	.336	1.71	.783
		.1112	.405	2.07	.784
		.1976	.551	2.81	.787
		.296	.668	3.41	.780
		.4420	.819	4.17	.783
5° Area Ratio 1 to 2	None	.0093	.130	.665	.854
		.0157	.176	.899	.886
		.0357	.265	1.35	.895
		.0503	.319	1.63	.903
		.0750	.432	2.20	.908
		.0880	.425	2.16	.904
		.0886	.425	2.16	.906
		.1700	.577	2.94	.895
		.2234	.671	3.42	.903
		.3520	.838	4.27	.899
10° Area Ratio 1 to 2	None	.0089	.130	.661	.875
		.0087	.132	.673	.899
		.0116	.146	.743	.869
		.0170	.183	.934	.893
		.0309	.250	1.28	.903
		.0378	.276	1.41	.902
		.0545	.333	1.70	.906
		.0582	.340	1.73	.893
		.0600	.347	1.77	.900
		.0656	.369	1.88	.915
		.0798	.400	2.04	.899
		.1005	.453	2.31	.908
		.1057	.465	2.37	.910
		.1880	.621	3.16	.910
		.2170	.672	3.42	.913
		.2540	.730	3.72	.920
		.2557	.722	3.68	.908
		.3090	.797	4.06	.910

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	h	q	v	c
90° Area Ratio 1 to 3 (Flush Entrance)	None	.0075	.104	.530	.766
		.0100	.123	.625	.778
		.0114	.131	.665	.775
		.0135	.144	.735	.783
		.0148	.151	.770	.793
		.0215	.185	.940	.799
		.0294	.224	1.14	.807
		.0310	.215	1.10	.795
		.0428	.257	1.32	.795
		.0535	.290	1.48	.798
		.0942	.389	1.98	.802
		.1727	.520	2.65	.798
		.1840	.540	2.76	.802
		.2636	.645	3.28	.802
		.3410	.739	3.76	.803
None	5° Area Ratio 1 to 2	.0172	.198	1.01	.961
		.0310	.271	1.38	.975
		.0313	.273	1.39	.976
		.0408	.311	1.58	.977
		.0489	.344	1.75	.986
		.0589	.378	1.92	.985
		.0700	.411	2.10	.987
		.0930	.473	2.41	.988
		.1060	.502	2.56	.980
		.1135	.524	2.67	.986
		.1155	.525	2.67	.978
		.1400	.594	3.03	1.01
		.1633	.623	3.17	.980
		.2560	.792	4.04	.994
		.3670	.948	4.83	.993
None	10° Area Ratio 1 to 2	.0182	.193	.985	.910
		.0361	.278	1.42	.930
		.0600	.362	1.85	.939
		.0807	.418	2.13	.932
		.1335	.543	2.76	.944
		.1560	.589	3.00	.945
		.1680	.606	3.08	.938
		.2930	.790	4.02	.934
		.4120	.960	4.89	.943

TABLE 2 (Continued)

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING CONICAL MOUTHPIECES

(Depth of Submergence, about 3½ Diameters)

Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	*h*	*q*	*v*	*c*	Entrance	Discharge	*h*	*q*	*v*	*c*
15° Area Ratio 1 to 2	None	.0161	.175	.893	.878	None	15° Area Ratio 1 to 2	.0120	.146	.743	.845
		.0288	.241	1.23	.902			.0185	.190	.968	.888
		.0450	.303	1.54	.906			.0230	.200	1.02	.861
		.0582	.350	1.78	.921			.0406	.281	1.43	.884
		.0633	.365	1.86	.909			.0444	.294	1.50	.885
		.0662	.405	2.07	.911			.0595	.344	1.75	.895
		.0852	.417	2.12	.906			.0610	.349	1.78	.900
		.1030	.463	2.36	.917			.0660	.367	1.87	.902
		.1827	.622	3.17	.926			.0670	.365	1.86	.904
		.2540	.728	3.71	.915			.0898	.428	2.18	.905
		.3520	.864	4.40	.925			.0891	.425	2.16	.901
		.4230	.952	4.85	.925			.1242	.500	2.55	.899
		.7650	1.25	6.40	.915			.1692	.578	2.95	.902
								.2637	.729	3.72	.893
								.2820	.746	3.80	.907
								.3400	.833	4.24	
20° Area Ratio 1 to 2	None	.0080	.123	.628	.877	None	20° Area Ratio 1 to 2	.0090	.123	.625	.821
		.0105	.147	.752	.914			.0148	.161	.821	.844
		.0191	.196	1.00	.905			.0153	.162	.827	.834
		.0306	.252	1.28	.912			.0160	.167	.849	.837
		.0355	.269	1.37	.908			.0244	.211	1.07	.859
		.0550	.338	1.72	.917			.0383	.269	1.37	.870
		.0695	.381	1.94	.917			.0404	.271	1.38	.856
		.0754	.394	2.01	.912			.0404	.272	1.38	.853
		.0846	.416	2.12	.908			.0437	.283	1.44	.860
		.1380	.540	2.75	.920			.0505	.305	1.55	.864
		.1736	.606	3.09	.924			.0747	.372	1.89	.863
		.2320	.707	3.60	.927			.0782	.385	1.96	.876
		.3075	.804	4.10	.921			.0953	.423	2.15	.870
		.4270	.964	4.91	.927			.1385	.514	2.62	.877
								.1867	.598	3.05	.879
								.2025	.626	3.19	.884
								.330	.795	4.05	.880
30° Area Ratio 1 to 2	None	.0080	.121	.619	.862	None	30° Area Ratio 1 to 2	.0096	.115	.586	.746
		.0083	.124	.633	.866			.0152	.146	.744	.751
		.0185	.187	.955	.874			.0152	.153	.779	.789
		.0221	.212	1.08	.907			.0176	.165	.841	.791
		.0367	.272	1.38	.900			.0291	.211	1.08	.786
		.0410	.287	1.46	.900			.0506	.281	1.43	.792
		.0470	.309	1.58	.906			.0519	.283	1.44	.789
		.0594	.347	1.77	.902			.0804	.359	1.83	.792
		.0804	.406	2.07	.911			.0852	.367	1.87	.798
		.0818	.414	2.11	.919			.1690	.513	2.61	.793
		.0967	.448	2.28	.915			.2331	.606	3.08	.797
		.1524	.563	2.87	.916			.2715	.657	3.35	.802
		.2404	.710	3.62	.921			.3990	.794	4.04	.798
		.3341	.838	4.27	.921						
		.390	.906	4.61	.923						
45° Area Ratio 1 to 2	None	.0189	.192	.975	.884	None	45° Area Ratio 1 to 2	.0078	.102	.521	.733
		.0278	.231	1.18	.885			.0163	.154	.787	.773
		.0404	.281	1.43	.888			.0340	.227	1.16	.785
		.0462	.304	1.55	.902			.0521	.282	1.44	.790
		.0721	.381	1.94	.902			.0546	.288	1.46	.782
		.0762	.386	1.97	.889			.0774	.343	1.75	.784
		.0965	.438	2.23	.896			.1185	.426	2.17	.785
		.1535	.557	2.84	.902			.2480	.617	3.14	.787
		.1857	.615	3.14	.907			.3970	.782	3.98	.789
		.325	.803	4.09	.890						
		.399	.902	4.59	.907						
		.451	.961	4.89	.910						

TABLE 2 (Continued)

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING CONICAL MOUTHPIECES

(Depth of Submergence, about 3½ Diameters)

Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	h	q	v	c	Entrance	Discharge	h	q	v	c
60° Area Ratio 1 to 2	None	.0079	.119	.606	.852	None	60° Area Ratio 1 to 2	.0116	.126	.644	.743
		.0083	.120	.614	.845			.0154	.149	.758	.762
		.0159	.174	.886	.876			.0396	.243	1.24	.776
		.0255	.219	1.11	.870			.0400	.246	1.25	.780
		.0383	.272	1.38	.883			.0577	.296	1.51	.780
		.0526	.316	1.61	.874			.0600	.304	1.55	.786
		.0553	.326	1.66	.877			.1035	.399	2.03	.787
		.0798	.394	2.01	.889			.1786	.526	2.68	.790
		.0800	.391	1.99	.880			.1940	.547	2.79	.792
		.1088	.465	2.37	.895			.2605	.642	3.27	.796
		.1437	.526	2.68	.882			.3101	.696	3.55	.795
		.2657	.716	3.65	.883			.4777	.859	4.38	.790
		.4129	.896	4.56	.886						
10° Area Ratio 1 to 3	None	.0080	.128	.654	.907	None	10° Area Ratio 1 to 3	.0103	.147	.750	.920
		.0150	.177	.900	.900			.0135	.168	.858	.924
		.0160	.179	.911	.899			.0185	.201	1.03	.941
		.0170	.185	.942	.901			.0304	.255	1.30	.931
		.0181	.190	.968	.897			.0319	.264	1.35	.940
		.0187	.196	1.00	.913			.0461	.317	1.61	.938
		.0198	.200	1.02	.902			.0530	.345	1.76	.956
		.0210	.210	1.07	.922			.0600	.360	1.83	.929
		.0500	.325	1 66	.922			.0749	.403	2.06	.934
		.0505	.325	1.66	.918			.0750	.404	2.06	.938
		.0516	.331	1.69	.924			.1000	.469	2.39	.951
		.0770	.398	2.03	.914			.1028	.479	2.44	.948
		.0960	.452	2.30	.928			.1378	.548	2.80	.938
		.1176	.497	2.53	.921			.1450	.572	2.92	.954
		.1570	.574	2.92	.921			.1740	.628	3.20	.956
		.1737	.609	3.11	.929			.1880	.662	3.37	.967
		.2375	.708	3.61	.925			.2175	.699	3.56	.954
		.2585	.745	3.80	.930			.2751	.794	4.04	.961
15° Area Ratio 1 to 3	None	.0080	.129	.660	.916	None	15° Area Ratio 1 to 3	.0090	.133	.675	.885
		.0165	.187	.955	.925			.0145	.170	.866	.896
		.0332	.257	1.31	.930			.0149	.169	.862	.880
		.0497	.324	1.65	.921			.0152	.172	.874	.880
		.0513	.330	1.68	.926			.0293	.242	1.24	.897
		.0778	.400	2.04	.912			.0588	.340	1.74	.893
		.1058	.476	2.34	.930			.0589	.345	1.76	.901
		.1200	.505	2.57	.927			.0812	.401	2.04	.896
		.1924	.638	3.25	.924			.1060	.466	2.38	.910
		.3634	.880	4.49	.928			.1238	.501	2.55	.905
								.1855	.610	3.11	.901
								.2043	.648	3.30	.914
								.2380	.705	3.59	.918
								.2685	.742	3.78	.910
								.2895	.777	3.96	.918
20° Area Ratio 1 to 3	None	.0080	.126	.644	.895	None	20° Area Ratio 1 to 3	.0080	.119	.607	.845
		.0140	.172	.874	.921			.0141	.160	.818	.867
		.0260	.234	1.19	.919			.0185	.182	.925	.848
		.0630	.366	1.87	.925			.0224	.204	1.04	.864
		.1010	.463	2.36	.924			.0225	.215	1.10	.855
		.1095	.483	2.46	.927			.0414	.278	1.41	.867
		.2101	.687	3.50	.930			.0440	.284	1.45	.872
		.3614	.880	4.49	.929			.0503	.310	1.58	.879
								.0855	.411	2.10	.892
								.1199	.486	2.48	.890
								.2029	.633	3.23	.892
								.2422	.694	3.54	.892
								.3071	.782	3.99	.897

TABLE 2 (Continued)

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING CONICAL MOUTHPIECES

(Depth of Submergence, about 3½ Diameters)

Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouthpiece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	h	q	v	c	Entrance	Discharge	h	q	v	c
30° Area Ratio 1 to 4	None	.0097	.144	.736	.932	None	30° Area Ratio 1 to 4	.0085	.122	.623	.841
		.0106	.149	.761	.920			.0206	.192	.980	.851
		.0306	.254	1.29	.922			.0241	.214	1.09	.874
		.0440	.306	1.56	.925			.0475	.297	1.51	.865
		.0599	.353	1.80	.918			.0584	.335	1.71	.880
		.0755	.397	2.02	.920			.0803	.394	2.03	.884
		.1079	.475	2.42	.920			.1112	.463	2.36	.882
		.1525	.570	2.91	.928			.1276	.500	2.55	.889
		.1870	.632	3.22	.930			.1600	.566	2.88	.899
		.1906	.638	3.25	.927			.2203	.666	3.39	.902
		.2801	.780	3.97	.936			.2789	.756	3.85	.910
								.2975	.777	3.98	.905
30° Area Ratio 1 to 3	None	.0085	.124	.630	.853	None	30° Area Ratio 1 to 3	.0076	.107	.546	.780
		.0165	.178	.909	.881			.0082	.106	.543	.770
		.0235	.220	1.12	.909			.0147	.150	.764	.785
		.0301	.247	1.26	.906			.0195	.173	.883	.788
		.0303	.247	1.26	.901			.0300	.215	1.09	.787
		.0643	.369	1.88	.925			.0310	.221	1.12	.796
		.0663	.374	1.91	.923			.0320	.217	1.11	.794
		.0950	.450	2.29	.928			.0520	.281	1.43	.795
		.1730	.611	3.11	.933			.0831	.360	1.83	.794
		.3053	.810	4.13	.932			.1454	.480	2.44	.800
								.2901	.682	3.47	.803
								.4600	.855	4.36	.801
45° Area Ratio 1 to 3	None	.0080	.119	.607	.845	None	45° Area Ratio 1 to 3	.0115	.131	.666	.773
		.0129	.153	.780	.853			.0178	.162	.825	.771
		.0232	.214	1.09	.893			.0305	.215	1.09	.779
		.0262	.231	1.17	.905			.0580	.297	1.52	.784
		.0412	.285	1.45	.890			.0993	.390	1.98	.785
		.0602	.346	1.76	.895			.1493	.475	2.42	.781
		.0783	.397	2.02	.901			.1970	.546	2.79	.782
		.1164	.490	2.50	.912			.2896	.669	3.41	.789
		.1256	.502	2.56	.900			.350	.737	3.76	.795
		.1801	.600	3.06	.898			.452	.836	4.26	.792
		.2233	.668	3.40	.896			.507	.917	4.67	.796
		.3117	.788	4.02	.897						
		.3223	.813	4.14	.906						
20° Area Ratio 1 to 2	5° Area Ratio 1 to 2	.0130	.232	1.18	1.29	20° Area Ratio 1 to 3	5° Area Ratio 1 to 2	.0091	.190	.968	1.27
		.0209	.301	1.54	1.32			.0221	.302	1.54	1.29
		.0280	.348	1.77	1.32			.0413	.416	2.12	1.30
		.0504	.464	2.37	1.32			.0626	.519	2.65	1.32
		.0520	.484	2.46	1.34			.1900	.686	3.49	1.35
		.0651	.539	2.75	1.34						
		.1083	.700	3.56	1.35						
		.1548	.836	4.26	1.35						
20° Area Ratio 1 to 2	10° Area Ratio 1 to 2	.0076	.158	.802	1.15	20° Area Ratio 1 to 2	10° Area Ratio 1 to 3	.0194	.261	1.33	1.19
		.0204	.263	1.34	1.17			.0274	.317	1.61	1.22
		.0397	.371	1.89	1.18			.0480	.345	2.16	1.23
		.0628	.470	2.35	1.18			.0734	.519	2.64	1.21
		.0915	.476	2.89	1.19			.1110	.639	3.26	1.22
		.1449	.708	3.60	1.18						
		.496	1.33	6.80	1.18						

TABLE 2 (Concluded)

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING CONICAL MOUTHPIECES

(Depth of Submergence, about 3½ Diameters)

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measused Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge	*h*	*q*	*v*	*c*	Entrance	Discharge	*h*	*q*	*v*	*c*
15° Area Ratio 1 to 2	15° Area Ratio 1 to 2	.0062	.128	.650	1.03	20° Area Ratio 1 to 2	15° Area Ratio 1 to 3	.0089	.160	.817	1.08
		.0192	.239	1.22	1.09			.0098	.170	.866	1.10
		.0433	.363	1.85	1.11			.0118	.196	1.00	1.14
		.0643	.443	2.25	1.11			.0224	.261	1.33	1.11
		.0690	.452	2.30	1.09			.0260	.292	1.49	1.15
		.3810	.975	4.97	1.09			.0456	.377	1.92	1.12
								.0479	.393	2.00	1.13
								.0624	.445	2.27	1.13
								.0890	.524	2.67	1.12
								.2022	.818	4.17	1.15
20° Area Ratio 1 to 3	20° Area Ratio 1 to 2	.0152	.191	.971	.984	20° Area Ratio 1 to 2	20° Area Ratio 1 to 3	.0388	.312	1.59	1.00
		.0333	.283	1.44	.985			.0651	.410	2.09	1.02
		.0621	.407	2.48	1.04			.0907	.494	2.52	1.04
		.0898	.486	2.07	1.03			.1434	.621	3.16	1.04
		.1512	.643	3.27	1.06			.1791	.695	3.55	1.04
		.233	.804	4.09							
20° Area Ratio 1 to 2	20° Area Ratio 1 to 4	.0343	.319	1.63	1.09	30° Area Ratio 1 to 2	10° Area Ratio 1 to 2	.0077	.160	.817	1.16
		.0559	.400	2.04	1.08			.0237	.282	1.44	1.17
		.0848	.481	2.46	1.05			.0340	.334	1.70	1.13
		.1233	.605	3.08	1.10			.0437	.381	1.94	1.16
		.1756	.726	3.70	1.10			.0630	.464	2.36	1.17
								.0650	.464	2.36	1.16
								.1860	.817	4.16	1.21
								.3030	1.04	5.30	1.20
								.500	1.37	6.97	1.23
30° Area Ratio 1 to 2	30° Area Ratio 1 to 2	.0227	.219	1.11	.921	45° Area Ratio 1 to 2	45° Area Ratio 1 to 2	.0108	.141	.721	.867
		.0625	.363	1.85	.922			.0301	.246	1.25	.901
		.0671	.381	1.94	.932			.0525	.330	1.68	.912
		.0719	.396	2.02	.940			.0633	.357	1.82	.900
		.1009	.469	2.37	.932			.0966	.443	2.25	.904
		.1560	.585	2.98	.944			.1411	.535	2.73	.905
		.1693	.608	3.11	.940			.2254	.681	3.47	.911
		.293	.796	4.06	.934			.466	.985	5.02	.912
		.373	.903	4.60	.932			.828	1.32	6.71	.915
		.465	1.07	5.49	.935						
		.742	1.27	6.48	.935						

TABLE 3

RESULTS OF EARLIER EXPERIMENTS ON CONVERGING MOUTHPIECES[1]

<table>
<tr><th rowspan="2">Source</th><th>Coefficient of Discharge Based on Smallest Area</th><th colspan="2">Mean Velocity at Smallest Area Feet per Second</th><th>Diameter or Side of Smallest Area</th><th>Ratio of Length of Straight Pipe Attached to d</th><th rowspan="2">Form of Mouthpiece</th></tr>
<tr><th>c</th><th colspan="2">v</th><th>d</th><th>$\frac{l}{d}$</th></tr>
<tr><td>Balch</td><td>.73 to .92</td><td colspan="2">4 to 15</td><td>4 in diam.</td><td>0.25 to 1.5</td><td>Conical, 45 degrees. Area Ratio 1 to 1.9</td></tr>
<tr><td>Davis and Balch</td><td>.829 to .955</td><td colspan="2">4.5 to 13</td><td>1.51 ft. sq.</td><td>0.65</td><td>Approach to each side of square was circular. Area Ratio 1 to 3. Discharge vertical</td></tr>
<tr><td>Stewart</td><td>.928 to .894</td><td colspan="2">1 to 4</td><td>4 ft. sq.</td><td>.077 to 3.5</td><td>Approach to each side of square was a quarter ellipse, the outer area being 8 feet square, the plane of which was 3 feet from entrance to straight pipe. Area Ratio 1 to 4</td></tr>
<tr><td>Ellis</td><td>.951 to .944</td><td colspan="2">12 to 32</td><td>1 ft. sq.</td><td>0</td><td>Approach to each side of square was a quarter ellipse with semi-diameters of 0.5 feet (vertical) and 0.83 feet (horizontal). Discharge vertical</td></tr>
<tr><td>Francis</td><td>.927 to .944</td><td colspan="2">5.5 to 9.2</td><td>1.22 in diam.</td><td>.082</td><td>Cycloidal. Area Ratio 1 to 1.95</td></tr>
<tr><td>Brownlee</td><td>.941 to .966</td><td colspan="2">7.5 to 27.2</td><td>0.1982 in. diam.</td><td>0</td><td>Cycloidal. Area Ratio 1 to 4</td></tr>
<tr><td>Weisbach</td><td>.959 to .994</td><td colspan="2">?</td><td>0.396 in diam.</td><td>0</td><td>Same as form of jet issuing from sharp edged orifice. Discharge into air</td></tr>
<tr><td></td><td></td><td colspan="2">Angles</td><td></td><td></td><td></td></tr>
<tr><td></td><td></td><td>Degrees</td><td>Minutes</td><td></td><td>Area Ratio</td><td></td></tr>
<tr><td rowspan="8">Castel</td><td>.924</td><td>5</td><td>26</td><td rowspan="5">0.6 in diam.</td><td rowspan="5">1 to 1.5
to
1 to 1.88</td><td rowspan="8">Conical. c varied but little with v. "Angle" means one-half total angle of convergence. $1/d$ = zero. There was contraction at exit. Discharge into air</td></tr>
<tr><td>.946</td><td>13</td><td>24</td></tr>
<tr><td>.924</td><td>19</td><td>24</td></tr>
<tr><td>.895</td><td>30</td><td>00</td></tr>
<tr><td>.870</td><td>40</td><td>20</td></tr>
<tr><td>.930</td><td>5</td><td>26</td><td rowspan="3">0.782 in diam.</td><td rowspan="3">1 to 1.7
to
1 to 10</td></tr>
<tr><td>.956</td><td>13</td><td>40</td></tr>
<tr><td>.920</td><td>35</td><td>52</td></tr>
</table>

[1] For reference to source see Appendix.

mental data in which each set of values represents, as a rule, the average of three experiments or runs although in some cases as many as six or eight experiments were made under practically the same head as explained above. In some cases, also, experiments were repeated at times differing by several days or even weeks. This was done mainly to check the work of different experimenters and to determine the effect of certain factors which caused trouble during the experimenting.

In Figs. 6, 7, 8, and 9 the experimental values of the coefficients of discharge for the short pipe with the various mouthpieces attached

TABLE 4

RESULTS OF EARLIER EXPERIMENTS IN WHICH DISCHARGE MOUTHPIECES WERE USED[1]

Source	Form and Angle of Entrance Mouthpiece or Approach		Area Ratio	Angle of Discharge Mouthpiece (One-Half of Total Angle)		Area Ratio	Coefficient of Discharge Based on Smallest Area	Diameter or Side of Smallest Section Inches	Connection
		Degrees[2]	$\frac{a}{A}$	Degrees	Minutes	$\frac{a}{A}$	c		
Weisbach	Sharp edge (no mouthpiece)		....	2	27	$\frac{1}{1.72}$	0.946	0.972	
Eytelwein	Circular	45	$\frac{1}{2}$	2	35	$\frac{1}{3}$	1.55	1.06	Connected without a straight pipe or throat
Francis	Cycloidal	45	$\frac{1}{1.95}$	2	30	$\frac{1}{2}$	1.55	1.22	Connected by a straight pipe or throat 0.1 inch long
Francis	Cycloidal	45	$\frac{1}{1.95}$	2	30	$\frac{1}{5.3}$	2.14	1.22	
Francis	Cycloidal	45	$\frac{1}{1.95}$	2	30	$\frac{1}{10}$	2.35	1.22	
Francis	Cycloidal	45	$\frac{1}{1.95}$	2	30	$\frac{1}{16}$	2.35	1.22	
Brownlee	Cycloidal	30	$\frac{1}{4}$	3 slightly	33 curved	$\frac{1}{22}$	2.20	0.1982	Connected without a straight pipe
Venturi	Rounded corners		?	2	43	$\frac{1}{2.7}$	1.46	1.33	Connected without a straight pipe
Davis and Balch	Circular	45	$\frac{1}{3}$	14+		$\frac{1}{1.78}$	1.03	12.51 (sq.)	Connected by a straight pipe 6 inches long. Vertical discharge
	Circular	45	$\frac{1}{3}$	curved		$\frac{1}{1.78}$	1.00	12.51 (sq.)	

[1] For references to source see Appendix. Velocities range about as in Table 3.

[2] Approximate.

have been plotted as ordinates, and the mean velocities in the pipe as abscissas. It will be noted that the mean velocity of flow through the pipe varies from about 0.5 ft. to 6 ft. per sec., although in many cases the upper limit was between 4 and 5 ft. per sec. These curves show that in all cases the coefficient of discharge is practically constant for velocities above 2 ft. per sec. and in some cases above 1 ft. per sec. However, for discharge mouthpieces having small angles of divergence the curves show a tendency for the coefficient of discharge to increase slightly with the velocity. This, it is thought, is due chiefly to the more turbulent condition of the water in the downstream compartment caused by these longer mouthpieces, which in turn made the measurement of the water level slightly too large. For the same reason the results for the longer discharge mouthpieces show the greater fluctuations. In all

TABLE 5

FINAL RESULTS FOR 6-INCH SUBMERGED SHORT PIPE HAVING MOUTHPIECES WITH AN AREA RATIO OF 1 TO 2 FOR VELOCITIES FROM 2 TO 5 FEET PER SECOND

(Depth of Submergence about 3½ Diameters)

Mouthpiece[1] Attached to Short Pipe, Area Ratio 1 to 2		Coefficient of Discharge from Experiment Based on Area of Pipe	Coefficient of Loss Based on Velocity Head in Pipe		Coefficient of Discharge Based on Outer Area of Mouthpiece	Coefficient of Loss Based on Outer Area of Mouthpiece $\left(\frac{1}{c_o^2}-1\right)$	Head Required on Pipe with No Mouthpiece Attached to Give Same Discharge[2] $H=\left(\frac{c}{.785}\right)^2 h$		Gain in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece (See col. 8, table 7)	Coefficient of Discharge for Discharge Mouthpiece when Entrance Mouthpiece is Attached $c_d = .785 \times \sqrt{\text{Col. 7 + Col. 8}}$	Per Cent Increase in *c* for Discharge Mouthpiece Due to Entrance Mouthpiece	Gain in Velocity Head Recovered by Discharge Mouthpiece Due to Entrance Mouthpiece, Expressed in Per Cent of Maximum Theoretical Amount Possible of Recovery. (See cols. 11 and 12, table 7)
Entrance Degrees	Discharge Degrees		$\left(\frac{1}{c^2}-1\right)$	Column 6 times $\left(\frac{a}{A}\right)^2$								
		c	m		c_o	m_o	H_E	H_D		c_d	Per Cent	Per Cent
1	2	3	4		5	6	7		8	9	10	11
0	0	.785	.620									
5	..	.900	.235				1.32h					
..	5	.990		.770	.495	3.08		1.59h	.51h	1.137	14.9	33.0
10	..	.910	.205				1.34h					
..	10	.940		.872	.470	3.50		1.44h	.19h	1.000	6.39	19.0
15	..	.920	.180				1.38h					
..	15	.900		.975	.450	3.90		1.32h	.11h	.938	4.22	7.9
20	..	.925	.165				1.39h					
..	20	.880		.104	.440	4.15		1.26h	.01h	.884	0.45	0
30	..	.920	.180				1.38h					
..	30	.800		1.31	.400	5.25		1.04h	0	.800	0	0
45	..	.900	.235				1.32h					
..	45	.790		1.37	.395	5.40		1.01h	0	.790	0	0
60	..	.885	.275				1.27h					
..	60	.790		1.37	.395	5.40		1.01h	0	.790	0	0

[1] Angle given is one-half total angle of convergence or divergence.

[2] This is called "equivalent head."

TABLE 6

FINAL RESULTS FOR A 6-INCH SUBMERGED SHORT PIPE HAVING MOUTHPIECES WITH AN AREA RATIO OF 1 TO 3 FOR VELOCITIES FROM 2 TO 5 FEET PER SECOND

(Depth of Submergence about 3½ Diameters)

Mouthpiece[1] Attached to Short Pipe Area Ratio 1 to 3 Entrance Degrees	 Discharge Degrees	Coefficient of Discharge from Experiment Based on Area of Pipe	Coefficient of Loss Based on Velocity Head in Pipe $\left(\frac{1}{c^2}-1\right)$	 Column 6 times $\left(\frac{a}{A}\right)^2$	Coefficient of Discharge Based on Outer Area of Mouthpiece	Coefficient of Loss Based on Velocity Head at Outer Area of Mouthpiece $\left(\frac{1}{c_e^2}-1\right)$	Head Required on Pipe with No Mouthpiece Attached to Give Same Discharge $H=\left(\frac{c}{.785}\right)^2 h$		Gain in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece (See col. 8, table 7)	Coefficient of Discharge for Discharge Mouthpiece when Entrance Mouthpiece is Attached $c_d = .785 \times \sqrt{\text{Col. 7 + Col. 8}}$	Per Cent Increase in c for Discharge Mouthpiece Due to Entrance Mouthpiece	Increase in Velocity Head Recovered by Discharge Mouthpiece Due to Entrance Mouthpiece, Expressed in Per Cent of Maximum Theoretical Amount Possible of Recovery. (See cols. 11 and 12, table 7)
		c	m		c_e	m_e	H_E	H_D		c_d	Per Cent	Per Cent
1	2	3	4		5	6	7		8	9	10	11
0	0	.785	.620									
10	..	.925	.165				1.39h					
..	10	.950		1.00	.317	9.00		1.47h	0.27h	1.035	8.95	14.7
15	..	.925	.165				1.39h					
..	15	.910		1.095	.303	9.86		1.32h	0.17h	.958	5.28	11.5
20	..	.925	.165				1.39h					
..	20	.890		1.14	.297	10.30		1.28h	0	.890	0	0
30	..	.925	.165				1.39h					
..	30	.800		1.44	.267	13.00		1.04h	0	.800	0	0
45	..	.900	.235				1.31h					
..	45	.790		1.49	.263	13.45		1.01h	0	.790	0	0
90	..	.800	.560				1.04h					
20 (1 to 4)	..	.925	.165				1.39h					
..	20 (1 to 4)	.900		1.23	.225	18.70		1.31h	0.04h	.912	1.33	9.0

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[1] Angle given is one-half total angle of convergence or divergence.

TABLE 7

FINAL RESULTS FOR 6-INCH SUBMERGED SHORT PIPE HAVING A MOUTHPIECE ATTACHED TO EACH END FOR VELOCITIES FROM 2 TO 5 FEET PER SECOND

(Depth of Submergence about 3½ Diameters)

Mouthpiece[1] Attached to Short Pipe		Coefficient of Discharge from Experiment Based on Area of Pipe	Same as Col. 3 Except Based on Outer Area of Mouthpiece	Coefficient of Loss Based on Velocity Head in Pipe $m_o = \left(\frac{1}{c_o^2}-1\right)\left(\frac{a}{A}\right)^2$	"Equivalent Head" or Head on Straight Pipe to Give Same Discharge $H_S = \left(\frac{c}{.785}\right)^2 h$	That Part of Equivalent Head Due to Discharge Mouthpiece $H'_D = \frac{H_S}{H_E}$	Increase in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece $H'_D - H_D$	Coefficient of Discharge Calculated from Data for Mouthpieces When Tested Separately $c' = .785 \times \sqrt{H_E \times H_D}$	Coefficient of Loss Based on c' but on Velocity Head in Pipe $m_e = \left(\frac{1}{c_o'^2}-1\right)\left(\frac{a}{A}\right)^2$	Ratio of Velocity Head Recovered to Maximum Possible of Recovery (in Per Cent)	
Area	Ratio									Contraction Suppressed	Contraction not Suppressed
Entrance Degrees	Discharge Degrees	c	c_o								
1	2	3	4	5	6	7	8	9	10	11	12
20 (1-2)	5 (1-2)	1.34	.720	.232	$2.92h$	$2.10h$	$0.51h$	1.17	.480	91.0	58.0
20 (1-2)	10 (1-3)	1.22	.407	.558	$2.42h$	$1.74h$	$0.27h$	1.12	.687	55.4	40.7
20 (1-2)	10 (1-2)	1.18	.590	.468	$2.26h$	$1.63h$	$0.19h$	1.11	.610	59.7	40.7
15 (1-2)	15 (1-2)	1.10	.550	.572	$1.97h$	$1.43h$	$0.11h$	1.06	.632	47.7	39.8
20 (1-2)	20 (1-3)	1.04+	.347	.811	$1.76h$	$1.28h$	$0.00h$	1.04	.811	26.6	26.6
20 (1-3)	20 (1-2)	1.04	.520	.672	$1.76h$	$1.27h$	$0.01h$	1.04	.672	32.4	32.4
30 (1-2)	30 (1-2)	0.940	.470	.875	$1.44h$	$1.04h$	$0.00h$	0.940	.875	7.35	7.35
45 (1-2)	45 (1-2)	0.910	.455	.955	$1.34h$	$1.02h$	$0.01h$	0.906	.962	4.00	4.00
20 (1-2)	20 (1-4)	1.075	.270	.795	$1.88h$	$1.35h$	$0.04h$	1.065	.880	33.0	24.0
20 (1-2)	15 (1-3)	1.13	.377	.675	$2.07h$	$1.49h$	$0.17h$	1.06	.777	42.0	30.5

[1] Angle given is one-half total angle of convergence or divergence.

cases, the coefficient of discharge decreases more or less rapidly as the velocity decreases from 1 or 2 ft. per sec.

Tables 5, 6, and 7 give the values of the coefficients of discharge and the corresponding values of the coefficients of loss for the short pipe with the various mouthpieces for velocities of from 2 to 5 ft. per sec. as taken from the curves in Figs. 6, 7, 8, and 9, together with certain other data used in the discussion which follows. Figs. 12 and 14 show the influence upon the action of discharge mouthpieces of attaching an entrance mouthpiece to the short pipe. The entrance mouthpiece suppresses the contraction at entrance to the pipe and allows the discharge mouthpiece to receive the water in a smoother state of flow than when the entrance end of the pipe is simply inward projecting. The influence of smooth flow is shown in Figs. 12 and 14 which are obtained by plotting the gain in rate of discharge and gain in velocity head recovered (expressed in per cent) as ordinates and the angle of discharge mouthpieces as abscissas.

8. *Inward-Projecting and Flush Entrance.*—The values of the coefficients of discharge for the short pipe with inward-projecting entrance (no mouthpiece attached) were determined with special care since the effect of attaching a mouthpiece could not otherwise be found. It will be noted from Fig. 6 and Tables 5 and 6 that the value of the coefficient of discharge and the coefficient of loss are respectively $c = 0.785$ and $m = 0.62$ while the values generally given in texts for an inward-projecting pipe are $c = 0.72$ and $m = 0.93$. That is, the head lost at the entrance to an inward projecting pipe is 0.62 of the velocity head in the pipe $(0.62 \frac{v^2}{2g})$ instead of $0.93 \frac{v^2}{2g}$ It would hardly be expected that all inward-projecting pipes would give the same coefficient of discharge, for such factors as the condition of the edge at entrance to the pipe, the diameter of the pipe or perhaps the ratio of the thickness of the pipe to the diameter, the degree of wetness of the material (effect of oil, etc.), the temperature and velocity of the water, the form and size of tank together with the location and form of piezometer orifice, and the conditions of discharge (submerged or into air) might easily influence the flow.

In order to get further data on this form of entrance another short pipe, 3.11 in. in diameter by 12 in. long, was tested in the same

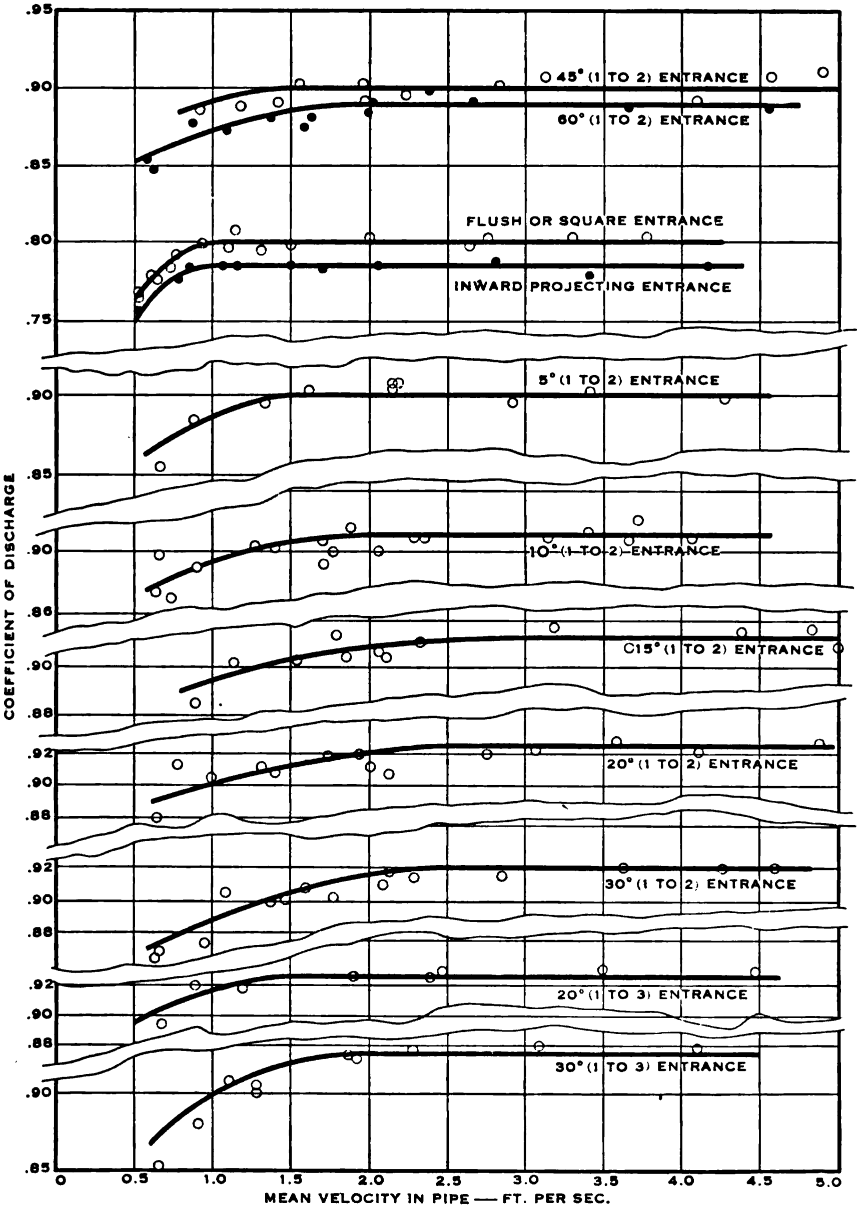

FIG. 6. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE

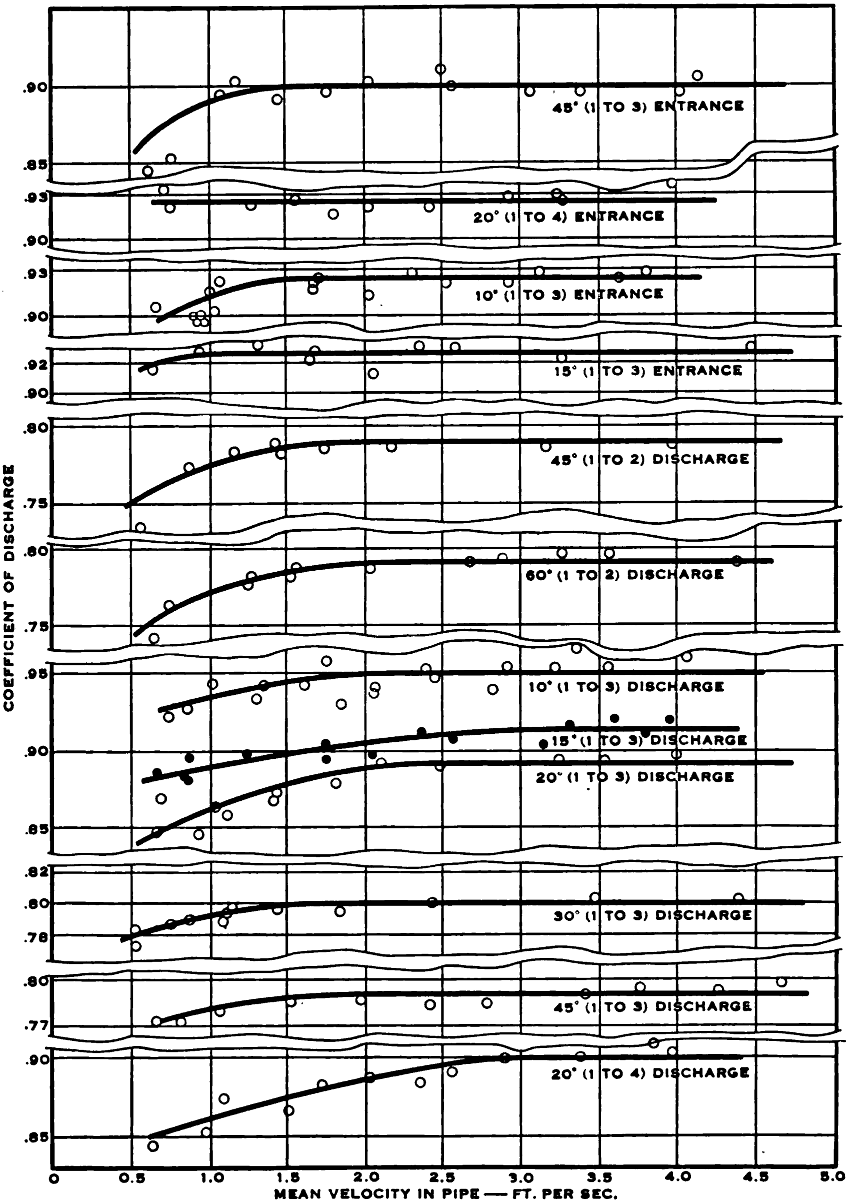

FIG. 7. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

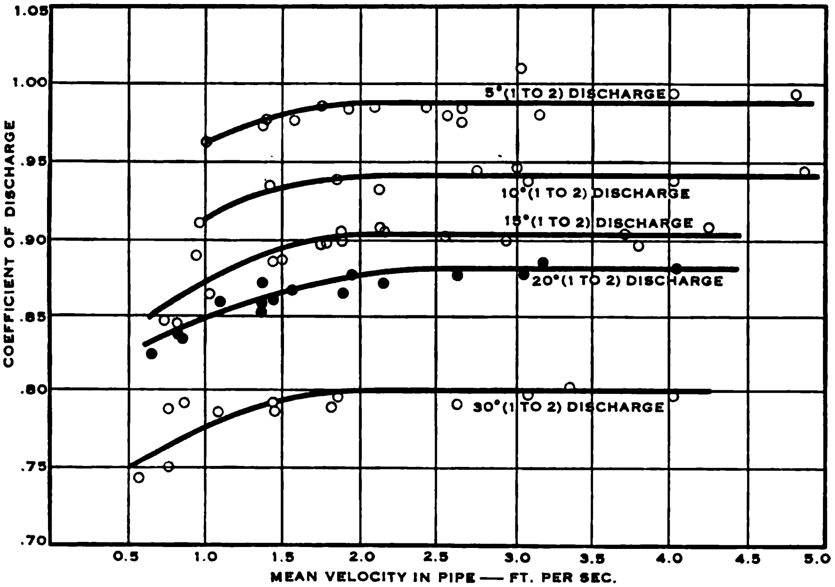

FIG. 8. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

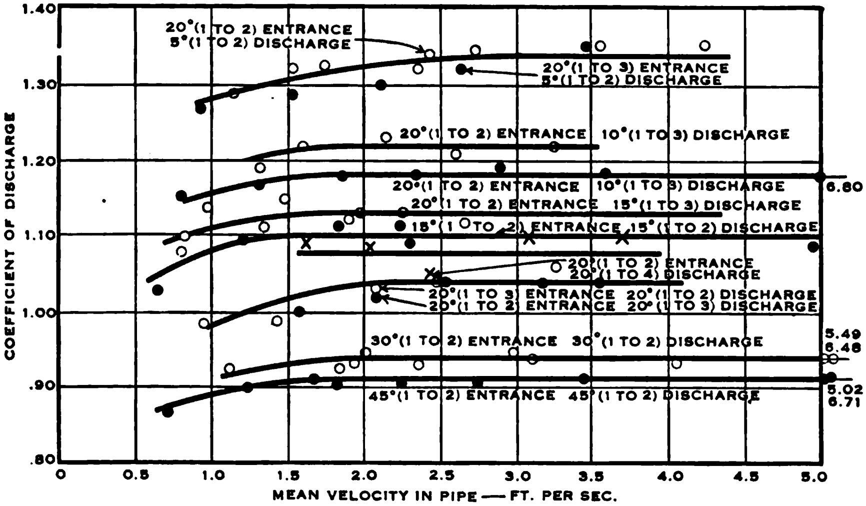

FIG. 9. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

tank, the maximum velocity being slightly greater than 5 ft. per sec. The values of the coefficient of discharge varied but little, ranging from 0.783 to 0.788. The ratio of the thickness of the pipe at entrance to the diameter for the 6-in. pipe was 0.05 and for the 3-in. pipe this ratio was 0.054. The entering edge of the 3-in. pipe was somewhat sharper than that of the 6-in. pipe.

The value for the lost head at entrance to an inward-projecting pipe as usually given seems to be based on rather meager data obtained chiefly from experiments with discharge into air at rather high velocities through tubes of small diameters. The value $c = 0.72$ for the coefficient of discharge seems to have been handed down from Weisbach. Bidone* reported a value of $c = 0.767$ and Bilton* a value of $c = 0.75$ for a 2½-in. pipe, increasing to 0.79 for a 1-in. pipe and to 0.93 for a ⅛-in. pipe, the length in each case being 2½ diameters.

The values of the coefficient of discharge and the coefficient of loss for a flush entrance (corresponding in these experiments to a 90-degree entrance mouthpiece having an area ratio of 1 to 3) also fail to check closely the values so generally given in texts and so generally used, namely, $c = 0.82$ and $m = 0.49$. As shown in Fig. 6 and Table 6, the values found in these experiments are $c = 0.80$ and $m = 0.56$. That is, the lost head at entrance to a short pipe having a flush entrance is $0.56 \frac{v^2}{2g}$. A 90-degree mouthpiece with an area ratio of 1 to 3 is ample to give the same conditions of flow as a reservoir wall which is flush with end of the pipe and hence it should give the same rate of discharge.

More experimental data on short pipes having flush entrances are available, and they cover a wider range of sizes and conditions than do those for inward-projecting pipes, the value of the coefficient of discharge ranging from 0.785 to 0.84 with the majority of the values lying below 0.82.*

The lost head at the entrance to a pipe can probably be determined better from a submerged tube than from one discharging into air. It would seem, therefore, that the value of the lost head at the entrance to an inward-projecting pipe is not so different from that for a pipe having a flush entrance as is usually believed, and that the common text book values need revision to apply to conditions commonly met in hydraulics.

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*See Appendix for references.

9. *Entrance Mouthpieces.*—From Tables 5 and 6 and the curves in Figs. 7 and 8, it will be seen that entrance mouthpieces having angles of from 10 degrees to 30 degrees (20 degrees to 60 degrees total angle of convergence) give practically the same discharge, while all the entrance mouthpieces having angles of from 5 degrees to 60 degrees (10 degrees to 120 degrees total angle) give only about 5 per cent range in the rate of discharge. In other words, the lost head at the entrance to an inward-projecting short pipe may be reduced from 0.62 of the velocity head in the pipe to 0.18 of the velocity head by a conical mouthpiece having an angle ranging from 10 degrees to 30 degrees. Also, the lost head will vary but little from 0.20 of the velocity head in the pipe for all entrance mouthpieces having angles between 10 degrees and 45 degrees (20 degrees and 90 degrees total angle). It should be stated, however, that conditions surrounding the entrance to the mouthpiece may have some effect, such as the accumulation of dirt in a passage or any other obstruction. Furthermore, it is clear from a comparison of Figs. 7 and 8 that no advantage results from increasing the length of the entrance mouthpiece beyond that corresponding to an area ratio of 1 to 2. The lost head at the entrance to a mouthpiece is sometimes considered the same as would occur at the entrance to an inward projecting pipe of the same area as that of the mouthpiece; that is, the lost head is found by multiplying the velocity head at entrance to the mouthpiece by the coefficient of loss for an inward projecting pipe. There seems to be little reason to justify such a method, since an entrance mouthpiece having an area ratio of 1 to 3 gives almost the same lost head as one with an area ratio of 1 to 2 while the velocity head at entrance to the former mouthpiece would be, of course, only one ninth of that of the latter.

It is not clear just what effect a straight throat or pipe has when added to an entrance mouthpiece. The discharge through the mouthpieces alone was not determined in these experiments. It would seem that if the mouthpiece suppressed the contraction very completely, the pipe would cause added resistance only, and hence decrease the discharge, while if the suppression was rather incomplete, the pipe might recover some of the velocity head during the expansion in it.

If this was in excess of the head lost during the expansion, the net results might be an increase in the discharge. Further, the suppression of the contraction by an entrance mouthpiece not only decreases the entrance loss but also reduces the turbulence of the subsequent flow in the pipe, thereby increasing the effect of a discharge mouthpiece. This will be discussed later under the heading of Combinations of Mouthpieces and Effect of Smooth Flow.

10. *Earlier Experiments with Entrance Mouthpieces.*—The results of other experimenters on entrance mouthpieces are not entirely consistent but give data of considerable importance. Table 3 gives in condensed form the results of the more important experiments. From this table it will be seen that adding a straight pipe to the mouthpiece used by Balch increased the discharge while in the experiments by Stewart the discharge was decreased. Futhermore, the coefficient of discharge increased with the velocity in the experiments by Balch and also in those by Davis and Balch but decreased in the experiments by Ellis, while Stewart found the coefficient to decrease at first and then to increase (not shown in Table 3). In the experiments recorded in this bulletin, the coefficient remained nearly constant. It will be observed that the mouthpieces used by the last four experimenters named in Table 3 have a throat diameter less than 1¼ in.; in fact in only one case is the throat diameter above 0.6 in. These mouthpieces give somewhat higher values for *c* than do mouthpieces of larger throat diameters. Perhaps the higher values for the coefficients of discharge for these small mouthpieces may be due to a lesser amount of turbulence; that is, it may be that the water flowing through a small mouthpiece is affected, or controlled more by the sides than in the case of a large mouthpiece, at least when the water enters or is received by the mouthpiece in a somewhat disturbed state of flow.

11. *Discharge Mouthpieces.*—As might be expected, the angle of the discharge mouthpieces influences the flow in a very different way from that of the entrance mouthpieces. Tables 5 and 6 and the curves of Figs. 7 and 8 show that when there is no mouthpiece on the entrance end of the short pipe, the coefficient of discharge for a discharge mouthpiece (attached to the pipe) diminishes rather rapidly as the angle of the mouthpiece increases, dropping somewhat

abruptly to practically no effect for an angle slightly greater than 20 degrees (40 degrees total angle).

It will be noted also by a comparison of Figs. 10 and 11 that increasing the length of the discharge mouthpiece beyond that corresponding to an area ratio of 1 to 2 has comparatively little effect on the rate of discharge. It may be, however, that an increase in length would show a greater effect for smaller angles than were used in these experiments, that is, for angles less than 5 degrees for an area ratio of 1 to 2 or less than 10 degrees for an area ratio of 1 to 3. But, it will appear (see article 12) that the size of the pipe, or the smallest area of the mouthpiece, is a much more important factor in the flow through a discharge mouthpiece than it is for an entrance mouthpiece. In other words, a discharge mouthpiece with a throat diameter of ½ in. may give quite different results from that of a discharge mouthpiece having the same area ratio and the same angle of divergence but with a throat diameter of 6 in., at least when the water is received by the mouthpiece in a turbulent state of flow. The discharge mouthpiece with the small throat diameter (having a relatively large ratio of circumference to cross-sectional area) seems to be able to affect a greater percentage of the water flowing and thus regain more energy per pound of water discharged. Furthermore, the chances of having the water approach the discharge mouthpiece with smooth flow is greater in the case of the small pipe, hence perhaps the two causes work together, and for any given case they would be difficult to separate. For these reasons a comparison with earlier experiments on discharge mouthpieces for the purpose of extending the present experiments is apt to be misleading.

It is clear from Figs. 10 and 11 that the governing factor in the recovering of velocity head by means of a discharge mouthpiece attached to a short pipe, having a diameter of several inches or more, is the angle at which expansion begins—rate of expansion at the start—at least when the total angle of divergence is not less than 10 degrees and the area ratio not less than 1 to 2.

12. *Earlier Experiments with Discharge Mouthpieces.*—Table 4 gives, in condensed form, the results of the more important earlier experiments with discharge mouthpieces. These experiments seem to show the influence of the size of throat area as discussed above. The increase in the coefficient of discharge, *c*, with an increase in

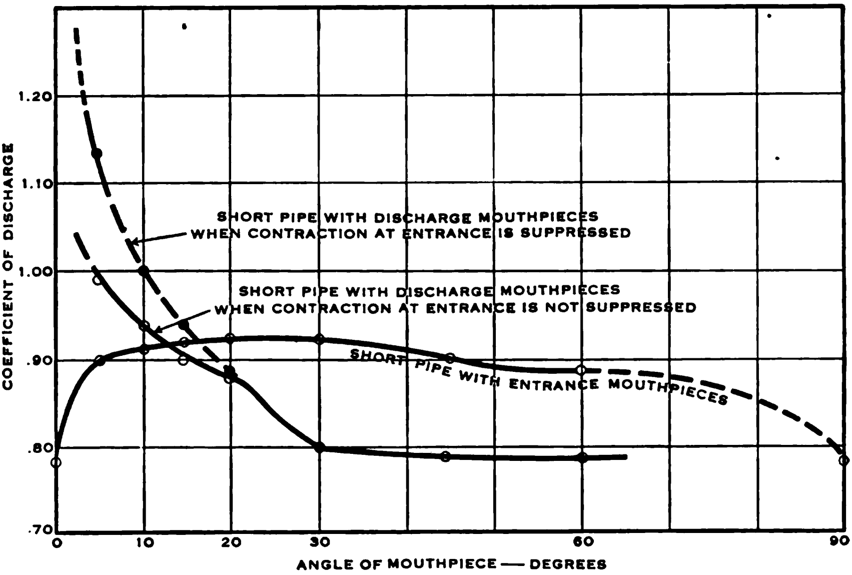

FIG. 10. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND ANGLE OF MOUTHPIECE FOR AREA RATIO OF 1 TO 2

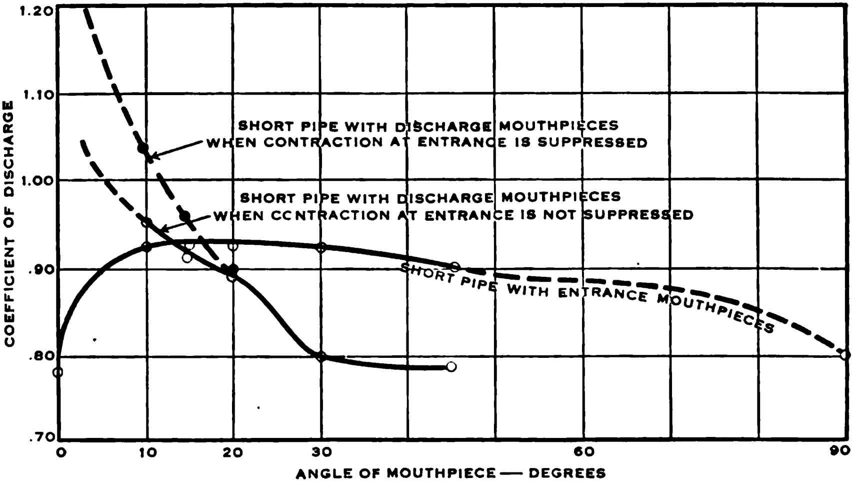

FIG. 11. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND ANGLE OF MOUTHPIECE FOR AREA RATIO OF 1 TO 3

the length of the discharge mouthpieces having small throat diameters is probably somewhat larger than would be obtained with mouthpieces having large throat diameters but with the same angles of divergence. As already noted in the experiments herein recorded the value of c increased very little for an increase in length corresponding to an increase in area ratio from 1 to 2 to 1 to 3 when the total angle of divergence was 20 degrees or more. It is clear from Table 4 that for discharge mouthpieces having small throat diameters and with the water in a state of smooth flow as it approaches the mouthpiece (entrance mouthpiece attached), an increase in length has a marked effect on the discharge. It will be noted that the greatest increase in c in the experiments reported by Francis occurred when the area ratio was increased from 1 to 2 to 1 to 5. It is probable that the increase in length would have been more noticeable in the present experiments for smaller angles of divergence, particularly for the smoother conditions of flow.

13. *Combinations of Mouthpieces and Effect of Smooth Flow.*—In order to get some measure of the influence of smooth flow upon the rate of discharge and the amount of velocity head recovered by a discharge mouthpiece, experiments were made using combinations of an entrance and a discharge mouthpiece when attached to the short pipe. The results are given in Table 7. It will be seen from this table that a discharge mouthpiece acts more effectively in recovering velocity head when a mouthpiece is attached to the entrance end. The following example will show this in the case of the 5-degree (1 to 2) discharge mouthpiece. This mouthpiece gave a coefficient of discharge of 0.99 (Table 5) when no mouthpiece was attached to the entrance end of the short pipe. From Table 7 it will be seen that the short pipe with a combination of the 5-degree (1 to 2) discharge mouthpiece and the 20-degree (1 to 2) entrance mouthpiece gave a coefficient of discharge of 1.34. This means that the 5-degree mouthpiece has a coefficient of discharge of 1.137 when used in combination with the 20-degree entrance mouthpiece. This value is obtained from the following steps: Attaching a 5-degree (1 to 2) discharge mouthpiece to the short pipe when no entrance mouthpiece is used is equivalent to raising the head on the inward projecting pipe from h to $1.59h$ as obtained by equating the rates of discharge,

$$0.99\ a\ \sqrt{2gh} = 0.785\ a\ \sqrt{2gH_D}, \text{ from which } H_D = 1.59h.$$

In like manner it is found that attaching a 20-degree (1 to 2) mouthpiece to the entrance end of the pipe when no mouthpiece is attached to the disharge end is equivalent to raising the head on the inward projecting pipe from h to $1.39h$ ($H_E = 1.39h$).

Now if both of these mouthpieces were attached to the short pipe, it might be expected that the head on the inward projecting pipe required to give the same discharge (equivalent head) would be $1.39h$ times 1.59 or $2.21h$ and that the coefficient of discharge for the combination, c_c, would be found from,

$$c_c\ a\sqrt{2gh} = 0.785\ a\sqrt{2g\ (2.21h)} \text{ or, } c_c = 1.17^{*} \quad . \quad . \quad (7)$$

As already noted, however, the coefficient of discharge for this combination as found from experiment is 1.34 (Table 7) which corresponds to an equivalent head of $2.92h$. If all of this increase is attributed to the more efficient action of the discharge mouthpiece due to the fact that it receives the water in more nearly parallel streamlines (smooth flow), the result is an equivalent head for the 5-degree (1 to 2) discharge mouthpiece of $2.10h$ (Col. 7, Table 7) instead of $1.59h$, an increase of $0.51h$ due to smooth flow. Hence, the coefficient of discharge for the short pipe with the 5-degree (1 to 2) discharge mouthpiece, assuming smooth flow as the water approaches the mouthpiece, would be found from

$$c_d\ a\sqrt{2gh} = 0.785\ a\sqrt{2g\ (2.10h)}, \text{ or } c_d = 1.137^{\dagger} \quad . \quad . \quad (8)$$

as compared with 0.99. In other words, this particular mouthpiece gives an increase, due to smooth flow, of 14.9 per cent in the rate of discharge.

Tables 5 and 6 give the values of the coefficient of discharge for the short pipe with the various discharge mouthpieces attached when the water flowed through an entrance mouthpiece on its way to the discharge mouthpiece. These values are represented by the dash lines in Figs. 10 and 11. Even though the entrance mouthpiece used gives stream lines that are far from parallel, it is thought that a less turbulent condition of flow would seldom be found at least in a 6-in. pipe, and for velocities above 2 ft. per sec. Hence, the values of the coefficient of discharge given by the dash lines are about the maximum to be expected. In each combination the entrance mouthpiece used was one giving about the minimum contraction.

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*See Table 7.

†See Col. 9, Table 5.

Tables 7 and 8 also give the percentage gain in the coefficient of discharge for the various discharge mouthpieces due to the entrance mouthpiece. The relation of this gain to the angle of the discharge mouthpiece is shown in Fig. 12 for both series of mouthpieces. It will be noted that as the angle of the discharge mouthpiece increases

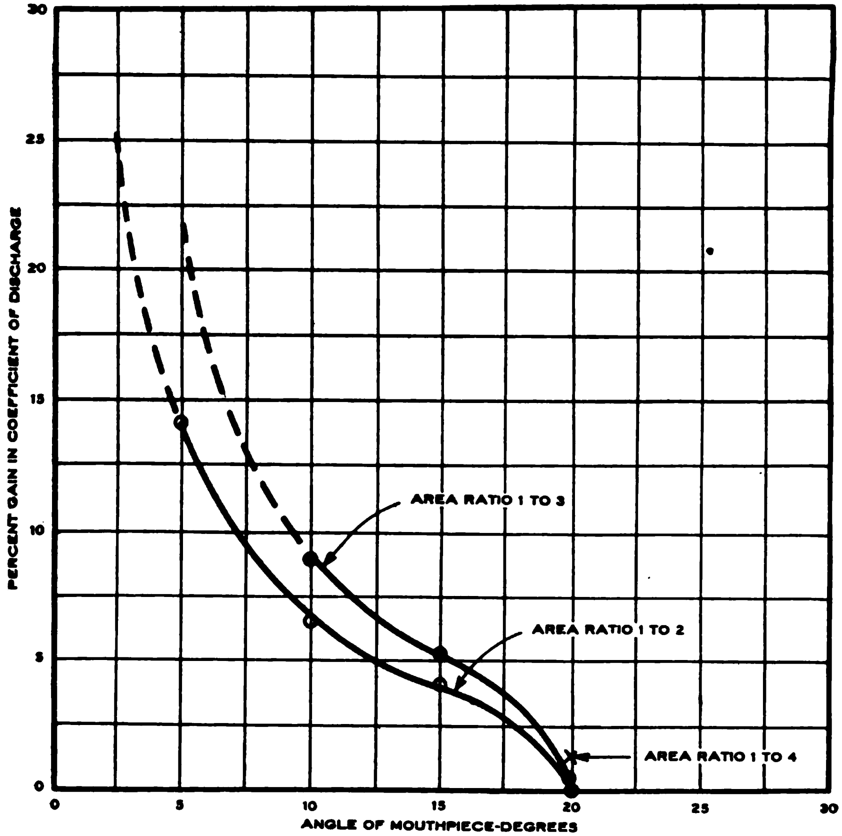

FIG. 12. RELATION BETWEEN PER CENT GAIN IN COEFFICIENT OF DISCHARGE FOR DISCHARGE MOUTHPIECES DUE TO ENTRANCE MOUTHPIECE, AND ANGLE OF DISCHARGE MOUTHPIECES

from 5 degrees (10 degrees total angle) the gain in the coefficient decreases rather rapidly, reaching zero for both series of mouthpieces at an angle of about 20 degrees (40 degrees total angle of divergence). Furthermore, this is the same angle at which a discharge mouthpiece rather abruptly ceases to regain any velocity head when no mouthpiece is used on the entrance end (see Figs. 10 and 11). It would seem, therefore, that a 20-degree discharge mouthpiece (total angle of divergence of 40 degrees), or one with a greater angle, allows such a turbulent condition of the water to develop that it is unable to recover any velocity head no matter how smooth the flow may be as the water approaches the mouthpiece. It is worth noting also that a comparison of the results in Table 5, Col. 10, and Table 6, Col. 10, indicates that the length of a discharge mouthpiece has a somewhat larger influence on the discharge when smooth flow exists than when the flow is more turbulent, and it is probable that this effect would have been more noticeable for smaller angles of divergence.

Attention has been called to the fact that the theoretical amount of velocity head which is possible of recovery by discharge mouthpieces having area ratios of 1 to 2, 1 to 3, and 1 to 4 is respectively 75, 88, and 94 per cent of the velocity head in the pipe. The effect of smooth flow may also be measured in terms of the increase in the amount of velocity head recovered by the discharge mouthpieces. For example, the coefficient of loss for the particular combination discussed above is 0.232 (Col. 5, Table 7) and the coefficient of loss for the pipe with the 20-degree (1 to 2) entrance mouthpiece only, is 0.165. Hence the loss of head in the discharge mouthpiece alone is 0.067 times the velocity head in the pipe. But 0.75 of the velocity head is the maximum amount possible of recovery, and since the mouthpiece lost 0.067 velocity heads, the amount recovered is 0.683 velocity heads which is 91 per cent of the maximum amount possible of recovery (Table 7, Col. 11). By a similar analysis it is shown that these same mouthpieces would have regained 58 per cent of the theoretical amount possible of recovery if the 5-degree (1 to 2) discharge mouthpiece had given the same action when in combination as it did when it was the only mouthpiece attached. That is, smooth flow allows a 5-degree (1 to 2) discharge mouthpiece to recover 33 per cent more velocity head than when the water approaches this mouthpiece in a turbulent state of flow (Table 5, Col. 11). The relation between the increase in the velocity head recovered by the discharge mouthpieces due to an entrance mouthpiece, expressed in per cent of the maximum theoretical amount possible of recovery, and the angle of the discharge mouthpieces is shown in Fig. 11. It is probable that the two curves should have the same ordinate for a 15-degree mouthpiece. At least, the curve for the mouthpieces having an area ratio of 1 to 3 should not be above the other curve. If the same entrance mouthpiece had been used in each case, the difference would probably have been negligible. The curves, therefore, are drawn to give the same ordinate for the 15-degree mouthpiece.

In the foregoing discussion it has been assumed that adding a discharge mouthpiece to the short pipe will not change the state of flow in the pipe. This is probably true if the pipe has an entrance mouthpiece attached, suppressing the contraction. If, however, there is no mouthpiece on the entrance end, it appears that the influence of the discharge mouthpiece extends back into the pipe, producing, in effect, a mouthpiece whose smallest cross-sectional area is somewhat less than the area of the pipe and perhaps allowing the expansion of the stream in the pipe to be continuous with that in the mouthpiece. For example, the short pipe with a 10-degree (1 to 2, 20-degree total angle) discharge mouthpiece gave a coefficient of loss of 0.872 (Table 5), and since the short pipe without any mouthpiece attached gives a coefficient of loss of 0.62, the mouthpiece alone should give a loss of 0.252 times the velocity head in the pipe. But from Table 7 it will be seen that this same mouthpiece gives in combination with a 20-degree (1 to 2) entrance mouthpiece a coefficient of loss of 0.468, and after deducting 0.165, which is the loss for the short pipe with the 20-degree entrance mouthpiece, 0.303 is left as the coefficient of loss for the discharge mouthpiece alone. This is inconsistent with the value, 0.252, already established. A similar effect occurs with all the mouthpieces except the 5-degree. The explanation, as already suggested, seems to be that the coefficient of loss for the short pipe is less than 0.62 when a discharge mouthpiece is attached; that the influence of the mouthpiece is felt back into the pipe. This also helps to explain why the coefficients of discharge for mouthpieces with small throat diameters and connected by a short throat are relatively large.

Fig. 13 shows the relation between the coefficient of loss for the short pipe with entrance mouthpiece attached and the angle of mouthpiece. It also shows the relation for the discharge mouthpieces alone as obtained by deducting the loss up to the mouthpiece. The values for the discharge mouthpieces alone are the larger ones as indicated in the example given. The values are given in Table 5. The number of velocity heads in the pipe recovered by any discharge mouthpiece would be the value obtained by subtracting the ordinate to the curve in Fig. 11 from 0.75 for an area ratio of 1 to 2 and from 0.88 for an area ratio of 1 to 3.

14. *Experimental Difficulties Encountered.*—During the early part of the investigation great difficulty was experienced in getting consistent results at the higher velocities, especially with mouthpieces having small angles of divergence. The head would fluctuate at times for apparently no good cause, and the value of the coefficient of discharge would vary considerably in successive experiments. It was noticed finally that vortices sometimes formed in the upstream compartment causing a "suck hole" as large as $1\frac{1}{2}$ in. in diameter at the water surface and tapering to a fine point some 10 or 12 in. below the surface. It was observed also that this vortex motion was more active and persistent when the unsteady conditions prevailed, but in no case did it appear to allow air to enter the pipe. In order to

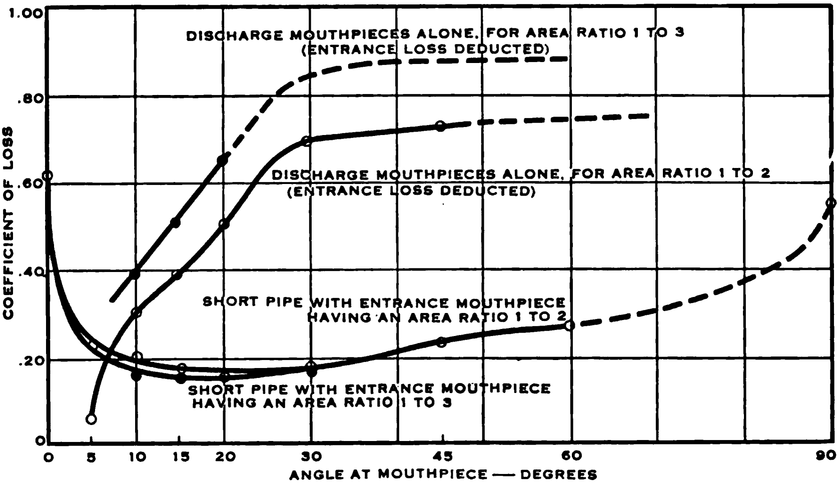

FIG. 13. RELATION BETWEEN COEFFICIENT OF LOSS AND ANGLE OF MOUTHPIECE

prevent the formation of the vortex a float was made fitting closely in the upstream compartment. Grill work on the under part of the float extended deep enough to suppress the vortex before it could fully form. Results were more consistent after the float was used, and it was kept in use for the remainder of the experiments. It was at once noticed, however, that the coefficient of discharge for the longer discharge mouthpieces was lowered by the use of the float and considerable time was spent in attempting to measure the effect of the vortex. The experimental results clearly indicated that the vortex motion may increase the discharge by at least 2 per cent in the case of the discharge mouthpieces with the smaller angles. If air had entered the pipe, the discharge would, of course, have been decreased. It would appear, therefore, that the whirling motion of the water allowed it to enter the pipe with less contraction. With a 15-degree discharge mouthpiece, the vortex seemed to show no effect although

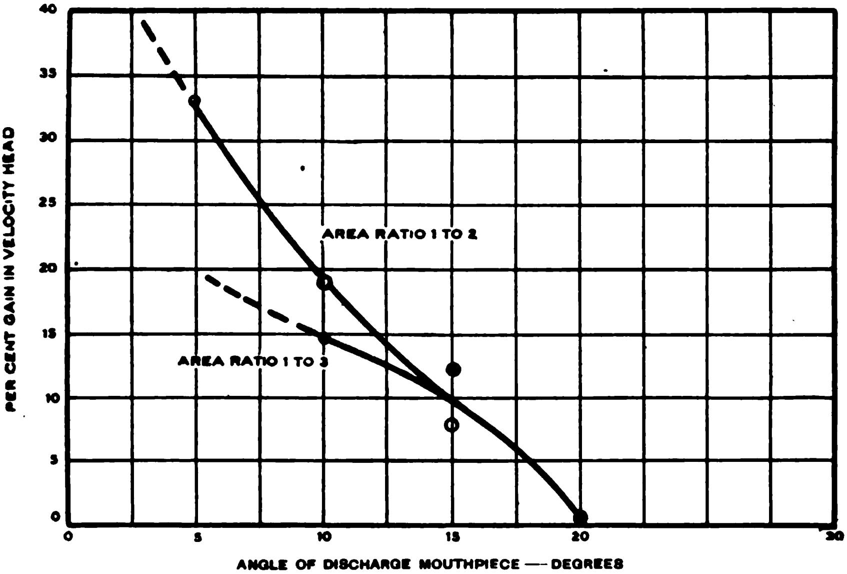

FIG. 14. RELATION BETWEEN GAIN IN VELOCITY HEAD RECOVERED BY DISCHARGE MOUTHPIECES DUE TO ENTRANCE MOUTHPIECE, EXPRESSED IN PER CENT OF THEORETICAL AMOUNT POSSIBLE OF RECOVERY, AND ANGLE OF DISCHARGE MOUTHPIECE

the vortex was not so large nor active as for the 5 and 10-degree mouthpiece. This suggests that perhaps the increase due to the vortex is caused more by the smoother flow of the water as it approaches the discharge mouthpiece, resulting in more efficient action by the mouthpiece (as already discussed), than by reduction in the entrance loss.

Another troublesome factor met with during the experiments was that of temperature changes of the water in the 2-in. pipes used as still basins for the hook gauges. The zero readings of the gauges were taken frequently during the experiments and were found to remain constant until the summer months when results became very erratic. After considerable time it was discovered that the trouble was due to the fact that the sun shone only on the pipe attached to the downstream compartment early in the afternoon, later shifting so as to shine only on the pipe attached to the upstream compartment. After correcting for the difference in temperature thus caused in the two still-basin pipes, consistent results were obtained. It was found, however, that the whole trouble could be avoided by frequently taking water from the tank and pouring it down the pipes.

15. *Conclusions.*—The preceding discussion has shown that the losses accompanying the flow of water depend largely upon the state of its motion which in turn is influenced by many factors, the effects of which in many cases can be but roughly estimated. While the results of these experiments tend to define the range of such effects for certain conditions of flow, additional experiments would be necessary to establish all the inferences which have been suggested. The following conclusions, however, seem justified:

a. As applying to conditions likely to be met in engineering practice, the value for the head lost at the entrance to an inward-projecting pipe (i. e. without entrance mouthpiece and not flush with wall of the reservoir) is 0.62 of the velocity head in the pipe ($0.62\frac{v^2}{2g}$) instead of $0.93\frac{v^2}{2g}$, as usually assumed. To put it in another form, the coefficient of discharge for a submerged short pipe with an inward-projecting entrance is 0.785 instead of 0.72 as given in nearly all books on hydraulics. Further, the lost head at the entrance to a pipe having a flush or square entrance is 0.56 of the velocity head in the pipe ($0.56\frac{v^2}{2g}$)

instead of $0.49 \frac{v^2}{2g}$ as usually assumed. In other words, the coefficient of discharge for a submerged short pipe with a flush entrance is 0.80 instead of 0.82 as given by nearly all authorities.

b. The loss of head resulting from the flow of water through a submerged short pipe when a conical mouthpiece is attached to the entrance end, may be as low as 0.165 of the velocity head in the pipe $(0.165 \frac{v^2}{2g})$ if the mouthpiece has a total angle of convergence between 30 and 60 degrees and an area of ratio of end sections between 1 to 2 and 1 to 4 or somewhat greater. In other words, the coefficient of discharge for a submerged short pipe with an entrance mouthpiece as specified above is 0.915.

c. The loss of head which occurs when water flows through a submerged short pipe having an entrance mouthpiece varies but little with the angle of the mouthpiece if the total angle of convergence is between 20 and 90 degrees and if the area ratio is between 1 to 2 and 1 to 4 or somewhat more. The loss of head for any mouthpiece within this range would be approximately 0.20 of the velocity head in the pipe $(0.20 \frac{v^2}{2g})$. There is, therefore, little advantage to be gained by making an entrance mouthpiece longer than that corresponding to an area ratio of 1 to 2. Thus, an entrance mouthpiece with a total angle of convergence of 90 degrees and the length of which is only 0.2 of the diameter of the pipe gives approximately $0.20 \frac{v^2}{2g}$ for the loss of head.

d. The amount of velocity head recovered by a conical mouthpiece when attached to the discharge end of a submerged short pipe depends largely upon the angle of divergence of the mouthpiece, but comparatively little upon the length of the mouthpiece. This is true for lengths greater than that corresponding to an area ratio of 1 to 2 and for total angles of divergence of 10 degrees or more. The amount of velocity head recovered decreases rather rapidly as the angle of divergence increases from a total angle of 10 to 40 degrees. At or near 40 degrees the amount of velocity head recovered rather abruptly falls to approximately zero.

e. A conical discharge mouthpiece having a total angle of divergence of 10 degrees and an area ratio of 1 to 2, when attached to a submerged short pipe, will recover 0.435 of the velocity head in the pipe, which is 58 per cent of the theoretical amount possible of recovery.

f. The amount of velocity head recovered by a diverging or discharge mouthpiece when attached to a submerged short pipe is considerably more when a converging or entrance mouthpiece is also attached than it is when the entrance end of the short pipe is simply inward-projecting (no mouthpiece attached). This excess in the velocity head recovered diminishes rather rapidly as the angle of the discharge mouthpiece increases, and it becomes zero for a discharge mouthpiece having a total angle of divergence of approximately 40 degrees. This increase in the velocity head recovered is probably due to the effect of smooth flow in the pipe as the water approaches the discharge mouthpiece. The smooth flow allows the mouthpiece to recover more of the velocity head in the pipe than when a more turbulent flow exists; this increase amounts to as much as 33 per cent in the case of the discharge mouthpiece having a total angle of divergence of 10 degrees and an area ratio of 1 to 2.

While these conclusions are drawn from experiments on the flow of water through a particular short pipe having various entrance and discharge conditions, it is felt that the results of the experiments are applicable in a general way to a large variety of cases in engineering practice where the contraction and expansion of a stream of water occurs. A number of such cases are suggested in the introduction.

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*This reference was not found until after this bulletin had been written. The sizes of the mouthpieces and their angles of divergence are so small (about 0.6 in. throat diameter and 4 to 12 degrees total angle of divergence) and the velocity is so high (32 to 131 ft. per sec. throat velocity) that the results are not directly comparable with the results given in this bulletin. The article contains, however, much valuable information, emphasizing in particular the decrease in lost head when the water is caused to whirl or rotate about the axis of the tube as it enters the tube. The above article also refers to experiments of Banninger, published in "Zeitschrift für das Gesamte Turbinenwesen," 1906, page 12. The results of these experiments have not been available for examination.

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EFFECTS OF STORAGE UPON THE PROPERTIES OF COAL

BY

S. W. PARR

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BULLETIN No. 97 MAY, 1917

EFFECTS OF STORAGE UPON THE PROPERTIES OF COAL

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PROFESSOR OF APPLIED CHEMISTRY

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ENGINEERING EXPERIMENT STATION
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EFFECTS OF STORAGE UPON THE PROPERTIES OF COAL

I. Introduction

The need of a thorough understanding of the conditions affecting the storage of bituminous coal is becoming more and more apparent. The demand for coal at certain seasons is so great that both mining and transportation facilities are taxed severely in meeting it. Provision has not been made for adequate and proper storage of bituminous coal either at the mines or at the distributing centers, and as a result of this lack there must be maintained throughout the year a sufficient number of operating mines to meet what may be termed the "peak load" which occurs during the winter months. At such times also there is often a shortage of cars, although a smaller number of cars even than is now available would be needed if the work of transportation could be more evenly distributed throughout the year. Mr. C. G. Hall, Secretary of the International Railway Fuel Association, has made an estimate showing that the number of excess mines over and above those which would be normally required to meet the demand, provided their work could be distributed evenly through the year, represents an investment in the United States of $450,000,000. In addition, the extra coal cars, which must be at hand when the demand is heavy but which stand idle accumulating rust for the rest of the year, have cost the railways no less than $105,000,000 more than would be necessary if the same tonnage could be hauled at an even rate. These wastes eventually affect the cost per ton of coal, which every one must pay as a contribution toward the capital investments.

The difficulties attending these conditions are accentuated by occasional abnormal demands such as are created upon the approach of any date for readjusting the wage scale with the accompanying possibility of a strike or lockout. For example, in addition to the normal excess demand during the winter months of 1915-16, a compilation of the published amounts of coal being stored by the various railway systems and larger users only, in view of a possible strike in April, aggregated over 3,000,000 tons.

The industrial disturbances do not include all of the serious considerations, however. If we consider the labor distresses that are accentuated as a result of irregular employment, it at once appears that the problems involved are of great sociological as well as of economic interest.

The work here recorded is a continuation of certain studies, the results of which have been published in bulletin form by the Engineering Experiment Station under the titles of "The Weathering of Coal,"* "The Occluded Gases in Coal,"† and "The Spontaneous Combustion of Coal."‡

The subjects treated in these publications are of fundamental importance, and a thorough understanding of the principles involved under each subject is necessary before any adequate discussion can be undertaken of the problems connected with the storage of coal. The investigations described in Bulletin 38 of the University of Illinois Engineering Experiment Station, on "The Weathering of Coal" were conducted with car-lot samples of coal stored under various conditions. The data obtained covered a period of one year. The investigation was continued for an additional period of five years more, or for a total period of six years. The coal was then turned over to the power plant for steam generation, and boiler tests were made to establish the character of the various samples. These data, together with other facts bearing upon the general subject of coal storage, have accumulated to an extent which warrants their being brought together for record and discussion in this form.

Special acknowledgment is due Mr. J. M. Lindgren, Chemist, and Mr. F. H. Whittum, Assistant Chemist, for the analytical data accumulated beyond the first-year period. Their experience and skill in the matter of sampling and in the use of analytical and calorimetric methods have been especially noteworthy.

II. Summary of Results

The facts established by this investigation may be briefly summarized as follows:

(1) Freshly mined coal is chemically very active. Certain constituents have a marked affinity for oxygen, with which they enter into combination at ordinary temperatures. While the extent of this reaction depends upon the variety of the coal and

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*S. W. Parr and W. F. Wheeler, Univ. of Ill. Eng. Exp. Sta. Bul. 38, 1909.
†S. W. Parr and Perry Barker, Univ. of Ill. Eng. Exp. Sta. Bul. 32, 1909.
‡S. W. Parr and F. W. Kressman, Univ. of Ill. Eng. Exp. Sta. Bul. 46, 1910.

the amount of these active constituents, a very important factor is the fineness of division or the sum total of the superficial areas of the particles, and accessibility of oxygen to the mass.

(2) The actual loss of heat value resulting from storage is small. It is evident that upon mining out the coal from the bed certain volatile constituents of the marsh gas variety are set free. The heat values represented by such exudations are not great. The tendency to absorb oxygen from the air is also a marked characteristic of freshly mined coal. This is in reality a chemical process, and is accompanied by the generation of a small amount of heat, but these heat losses, compared with the total heat available in the coal, are insignificant. Indeed, it may be fairly questioned whether the heat losses are not more apparent than real since there is an increase of weight due to the absorption of oxygen. Such increase will in itself lower to a corresponding degree the indicated heat value per pound of coal.

(3) There is an increase of ''fines'' or slack resulting from storage, greater with some coals than with others. This, together with the saturation of the free burning constituent with oxygen, slows up the fire and gives the appearance of being lacking in heat value. However, with an increase of draft and a correct understanding of the combustion conditions to be maintained, a most excellent over-all efficiency can be secured even from coals which have been in storage for long periods.

(4) Bituminous coal can be stocked without appreciable loss of heat values provided the temperature is not allowed to rise above 180 degrees F. Any method of storage, to be successful, must either check or prevent the absorption of oxygen to such an extent that the generation of heat shall not proceed faster than the dissipation and loss of heat due to absorption or radiation.

(5) Underwater storage prevents loss of heat values, and is not accompanied by deterioration in physical properties, such as slacking. The water retained by the coal upon removal is substantially only that held by adhesion or capillarity.

(6) Dry storage is safer and more satisfactory if the fine material is screened out at the storage yard and lump only, preferably sized, is stocked.

It will be seen from this summary that the most serious part of the problem relates to the matter of spontaneous heating, and probably the least serious phase relates to deterioration and actual loss of heat values. It is certain that at the present time a better understanding of these difficulties has been reached, and there is reason for believing that this better understanding of the fundamental principles involved will lead to some practicable and safe procedure for the stocking of bituminous coal.

III. Effects of Storage Upon Coal

HEATING AND SPONTANEOUS COMBUSTION

It is a well established fact that freshly mined coal has a large absorptive capacity for oxygen. In Bulletin 32, "The Occluded Gases in Coal," it is made evident that this avidity for oxygen is most marked in the freshly mined coal, and after exposure to the air for four or five months an approach to the saturation point seems to be reached after which very little oxygen is taken on. A correct interpretation of this phenomenon is essential to an understanding of the spontaneous heating of coal piles. The natural conclusion would be to the effect that the oxygen has been simply absorbed or occluded; that it was a physical rather than a chemical change. The evidence, however, of all the more recent investigations goes to show that it is in fact a chemical combination and that it is accompanied by the generation of a small amount of heat. In Bulletin 46, "The Spontaneous Combustion of Coal,"* it is seen that at a temperature of from 35 to 40 degrees C (95 to 104 degrees F), and with free access of air, the amount of heat generated caused a rise in temperature of from 1 to 1½ degrees C per day. Porter and Ovitz† have measured the quantity of oxygen taken up by a sample of Franklin County coal and found it to be approximately 0.8 per cent of the weight of the coal. Moreover, there is only a very small amount of CO_2 formed. This is explained by the fact that the presence of certain unsaturated compounds allows the oxygen to enter the molecular structure of the coal, with which compounds the oxygen readily combines.

These references are a few of many that might be brought forward showing that at ordinary temperatures freshly mined coal unites chemically with oxygen and that in the process there is generated a certain amount of heat.

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*Parr and Kressman, Univ. of Ill. Eng. Exp. Station Bul. 46, 1910.

†Journal Industrial and Engineering Chemistry, Vol. 2.

1. *Oxidation of Pyrites.*—Much consideration has been given by various investigators to the role of iron pyrites in promoting the heating of coal. In the experiments carried out by Dennstedt and Bunz in 1908,* it appears that self-ignition may be brought about in the case of coals having only small amounts of pyritic sulphur. The conclusion is made, therefore, that the presence of iron pyrites is not an essential condition for spontaneous heating. Other investigators working along similar lines have reached the same conclusion. Still others seem to have evidence that pyritic sulphur is an active element in the case. A summary of opinions on this point is given as follows:†

"As to what part sulphur compounds, especially pyrite, play in the spontaneous ignition of coal, opinions differ greatly. Some believe pyrite to be the leading factor, while others believe it plays no part at all, or, if so, ascribe to it a position of minor importance and believe its action to be merely a subsidiary one. The oxidizing action of the air upon pyrite is, however, admitted, and the notion seems to be fairly general and well established that pyritic oxidation tends to raise the temperature of the coal. On the other hand, it is seen from the work of Fayol, Dennstedt and Bunz, Threlfall and others that coals containing pyrite in a quantity too insignificant to be noticed are very apt to ignite spontaneously. The Newcastle coal of New South Wales is also a very good example of this class of coals. Others, however, believe that the only influence of the pyrite is a mechanical one, in which the oxidation of the thin films of pyrite in the coal serves merely to break up the coal."

Investigations of this type, having for their object a study of the processes of oxidation which occur at normal temperatures, are extremely important, especially in that phase of the work which seems to have fully established the fact of oxidation of the organic constituents of the coal. The conclusions are somewhat at fault, however, in assuming that as a consequence the pyritic oxidation is of little importance. It is true that a coal may heat seriously even though pyritic sulphur is absent. This does not constitute proof, however, that the presence of pyritic sulphur in coals may not be equally, or even more largely, responsible for heating than the organic constituents.

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*Zeit für Aug. Cheml., Vol. 21, pp. 1821-35, 1908.

†Parr and Kressmann, "The Spontaneous Combustion of Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 46, pp. 83, 1910.

This is clearly set forth in Bulletin 46.* On page 52, under "Iron Pyrites," a summarized statement is given as follows:

> "The presence of sulphur in the form of iron pyrites is a positive source of heat due to the reaction between sulphur and oxygen. This may be conveniently referred to as the second stage in the process of oxidation. Here again rapidity of oxidation is directly dependent upon fineness of division. Since coals, as a rule, have a much higher earthy or ash content in the fine duff, and since iron pyrites is a large component of this material, it follows that the presence of dust or duff in all coals of the Illinois type is a positive source of danger."

However, in this summary the authors give first place in time and effect to the oxidation of the organic matter and consider that the activity of the pyrites waits somewhat upon the rise in temperature from such organic oxidation before action with sulphur reaches a serious phase. Special emphasis was laid upon the oxidation of sulphur as a source of heat, but the experiments did not specifically give direct evidence as to the temperature at which pyritic sulphur began to oxidize.

2. *Growth of Sulphates.*—Data on the oxidation of sulphur have recently been developed in connection with the study of variations in the determination of ash values† which have a bearing in this connection. The fact appears that the oxidation of sulphur is active at ordinary temperatures provided (a) that the pyritic iron be finely divided, and (b) that free moisture be present in sufficient amount to satisfy the reactions involved. For example, a certain series of bed samples of coal had been ground to 60-mesh and laboratory samples taken of about 75 grams which were placed in 4-ounce bottles with rubber stoppers. These samples were retained in the laboratory at room temperature from August, 1912, to April, 1913, at which time they were analyzed for sulphur in the sulphate or SO_3 form. For comparison, the original samples ground only to 10-mesh were similarly analyzed. The results are shown in Table 1.

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*Parr and Kressmann, Univ. of Ill. Eng. Exp. Sta. Bul. 46, 1910.

†S. W. Parr, Ill. State Geo. Survey, Co-Op. Bul. 3, 1915.

TABLE 1

GROWTH OF SULPHATE

COMPARISON BETWEEN FINE AND COARSE LABORATORY SAMPLES

Lab. No.	County	H_2O	Total Sulphur	SO_3 60-Mesh Aug., '12	SO_3 60-Mesh April, '13	SO_3 10-Mesh April, '13
5365	Mercer.........	6.33	5.29	0.95	1.46	0.86
5367	Grundy........	8.84	2.27	0.39	0.38	0.18
5370	Mercer.........	4.49	4.92	0.47	0.49	0.25
5372	Mercer.........	4.86	5.46	0.63	1.25	1.12
5376	Grundy........	7.66	2.73	0.61	1.12	0.86
5388	La Salle........	7.93	5.20	1.42	1.79	0.82
5381	La Salle........	8.05	4.83	0.62	1.20	0.81

3. *Effect of Fineness of Division.*—From the results in Table 1 it is evident that oxidation of sulphur has occurred at ordinary temperatures. Five of the seven samples have had from 30 to 40 per cent of the total sulphur thus changed. The second and third samples in this table show no such oxidation. In the second the content of total sulphur is low and of this the actual sulphur in the pyritic form is, of course, still lower. Data on this point were not obtained. In the third sample the water content is lower than that in the other samples of the table. Whether this affords a valid explanation is uncertain. At any rate the point here emphasized is the fact that in the majority of the samples oxidation of the pyritic sulphur occurred in large amounts and at room temperatures. The next point was to determine what conditions were chiefly responsible in this reaction. The last column of the table affords some information. Here it is seen that with one exception the coarse or 10-mesh material had little or no indication of sulphur oxidation. To test further the effect of fineness of division, two of these original samples, about two pounds each, were sized and the sulphate sulphur determined for each size. The results are given in Table 2.

TABLE 2

SULPHATE SULPHUR AS FOUND IN VARIOUS SIZES OF COAL

Lab. No.	10-Mesh	20-Mesh	40-Mesh and Over
5372	0.91	0.89	1.43
5388	0.53	0.60	1.42

It is shown by this table that while substantially no increase in sulphate occurred up to the 20-mesh size, the increase was very marked in the 40-mesh sample, though it should be said that the 40-mesh sample contained also all of the finer material passing through that sieve. This record further emphasizes the fact that oxidation of sulphur increases in activity as the size of particles is decreased and the superficial area in any given mass correspondingly increased.

4. *Effect of Moisture.*—In seeking an explanation for the lack of uniformity in behavior due to sizing alone, it was thought that possibly the amount of free moisture in the sample as well as the percentage of FeS_2 might play an important part. A number of samples were, therefore, selected in which the free moisture was low. In these cases the growth of sulphate in the laboratory sample was small as shown in Table 3.

TABLE 3

EFFECT OF LOW MOISTURE ON THE FORMATION OF SULPHATE

All Values on the Dry Coal Basis

Lab. No.	Moisture	Total Sulphur	Sulphate (SO_3) in Lab. Sample, 60-Mesh 75 Days in Sample Bottle	SO_3 in 3-Lb. Gross Sample. All Sizes up to ¼ Inch after 15 Days in Container	3-Lb. Gross Sample. All Sizes up to ¼ Inch after 75 Days	Sizing of Gross Sample after 75 Days		
						SO_3 in 10-Mesh Size	SO_3 in 20-Mesh Size	SO_3 in 40-Mesh Size
6399	1.74	3.65	.214	.200	.198	.176	.164	.215
6400	2.03	3.19	.218	.230	.228	.199	.195	.257

As affording further evidence on this general proposition, 28 samples of coal were selected from the various districts of the state, and sulphate determinations made on the laboratory samples ground to 60-mesh which had been in storage from the early part of 1912 until June, 1913. Unfortunately, the sulphate factors for the fresh coal are not available, but the table shows that in those samples in which both the water and the sulphur contents were high, there was a greater increase in the percentage of oxidized sulphur than in the case of samples in which water and sulphur contents were low. Note especially Nos. 5359-5389, inclusive, Table 4.

Doubtless, there are other circumstances connected with the oxidation of sulphur which are of interest, such as the presence of catalyzers, the source of oxygen for satisfying the conditions of the

TABLE 4

SULPHATE IN LABORATORY SAMPLES, STORAGE TIME FROM MARCH, 1912, TO JUNE, 1913, SHOWING CONDITIONS ATTENDING PRESENCE OF MOISTURE AND SULPHUR

Lab. No.	County	Coal Bed	H_2O	Total Sulphur Dry Coal	SO_3 Dry Coal	Remarks
4699	Vermilion	6 N	2.30	2.44	0.18	Sulphate was determined June 2, 1913, on pulverized samples collected February to June, 1912.
4702	Vermilion	6 N	2.08	2.75	0.15	
4706	Vermilion	6 N	2.08	3.48	0.32	
4707	Vermilion	6 N	1.82	4.82	0.35	
4716	Vermilion	7 N	2.31	4.06	0.62	
4724	Vermilion	7 N	1.95	3.77	0.60	
4727	Vermilion	7 N	1.91	3.33	0.37	
4734	Vermilion	7 N	2.24	2.59	0.22	
4744	Vermilion	6 N	4.58	1.94	0.29	
4789	Franklin	6 S	3.88	0.52	0.02	
4811	Franklin	6 S	2.41	1.53	0.22	
4994	Saline	5 S	2.82	2.32	0.54	
5006	Williamson	6 S	3.25	1.11	0.12	
5011	Franklin	6 S	4.14	1.53	0.18	
5024	Saline	5 S	5.87	3.78	0.80	
5121	Williamson	6 S	3.73	1.42	0.11	
5122	Williamson	6 S	6.25	1.46	0.12	
5134	Williamson	6 S	6.25	1.24	0.10	
5224	Franklin	6 S	5.87	1.12	0.06	
5339	Mercer	1 N	3.21	5.66	0.98	
5359	Rock Island	1 N	4.84	6.56	2.14	
5361	Rock Island	1 N	5.74	4.56	1.08	
5362	Rock Island	1 N	4.96	5.26	1.15	
5364	Mercer	1 N	6.34	4.94	1.69	
5368	Grundy	2 N	7.08	3.04	0.59	
5369	Grundy	2 N	6.98	2.53	0.44	
5377	Grundy	2 N	7.83	4.00	1.34	
5389	La Salle	2 N	7.71	3.57	0.83	

reaction, the mercasite or pyrite form of the sulphide crystals, the method of distribution, whether in microscopic or massive aggregates, the character and influence of the associated material, and the segregation of sulphate crystals towards the finer materials. Indeed, the more thorough understanding of some of these points might contribute in a very practical way to our knowledge of the conditions which promote the heating of coal. Definite information along these lines depends upon further study. However, for purposes of this discussion it is of importance to note that at least two conditions, if existing, namely, fineness of division, and presence of moisture, will result in oxidation of the sulphur. Supplementary to this should be recalled the fact already developed in Bulletin 46 that the oxidation of 0.5 per cent of sulphur, or approximately less than ⅕ of the amount present in the average Illinois coal, would produce sufficient heat to raise the temperature of the mass, not allowing for radiation losses, about 125 degrees F. If the initial temperature were 50 degrees F, an increase to 175 degrees would approach the danger point. At about 180 degrees the activity reaches a stage owing to the greater rapidity of oxidation at that temperature, at which the chemical reaction quickly proceeds to the point where it becomes autogenous.*

In the discussion concerning the generation of heat from sulphur oxidation it is not intended to minimize the effect of oxidation of the organic matter as an initial source of heat, independent of the activity of the sulphur. In bituminous coals the two, doubtless, proceed independently, but where both activities exist together there is an acceleration of the reaction due to the rapid rise of temperature. In this manner there is a greater quantity of heat produced in a given period of time, and hence the coal mass comes more quickly and more positively to the autogenous or danger stage.

5. *Summary on Oxidation.*—Oxidation of the organic materials in freshly mined coal is active in all coals of the bituminous or lignitic type. The conditions which accelerate the action are increase of temperature and fineness of division.

Oxidation of the sulphur of iron pyrites is active provided the sulphide of iron is finely divided and there is sufficient moisture present to satisfy the reaction. The quantity of finely divided pyritic material will of course be greater in screenings than in lump, and since the ash content in screenings is from one to two times as great as in the lump, the quantity of pyritic sulphur is correspondingly greater.

The conditions to be observed in stocking coal so far as oxidation is concerned, are thus fairly well outlined. The enumeration of these conditions will be taken up later.

DETERIORATION

Experiments on the weathering of coal are described in Bulletin 38.† Car lots were stored in open and in covered bins and smaller lots of 100 to 200 pounds under water. The samples for determination of heat values were taken on the day of mining the coal, and thereafter at seven days, two months, six months and one year.

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*Note the abrupt change in the direction of the curves at 80 degrees C, showing temperature rise, as in "Spontaneous Combustion of Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 46, p. 28.

†Parr and Wheeler, "The Weathering of Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 38, 1909.

All of the samples with the exception of the submerged samples, were continued in storage beyond the period covered by the results presented in Bulletin 38, for a total period of six years. Two additional sets of laboratory samples were taken; one after a total period of three years, and another at the end of six years. After the storage in the original bins for four years, both the covered and open lots were moved to a new location. After a period of six years, final laboratory samples were taken in connection with the cleaning up of the various lots and the making of boiler tests. Five standard boiler tests were made at the close of the period.

For greater convenience all of the analytical results for the entire period are presented in Tables 5 to 10, inclusive.

TABLE 5

VERMILION COUNTY NUT COAL

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		F. t. u.	Per Cent
	STORED IN EXPOSED BINS						
1031	Same day	10.55	4.25	12991	14814	...	....
1081	7 days	13.98	2.65	12412	14716	98	0.66
1240	2 months	14.21	2.47	12265	14577	237	1.60
1656	6 months	13.53	2.10	12396	14575	239	1.61
2088	1 year	13.62	2.82	12282	14498	316	2.13
4105	3 years	13.22	1.75	12018	14075	739	4.98
7298	6 years	13.06	1.58	11984	14000	814	5.49
	STORED IN COVERED BINS						
1031	Same day	10.55	4.25	12991	14814	...	....
1081	7 days	13.98	2.65	12412	14716	98	0.66
1249	2 months	13.08	2.13	12475	14604	210	1.42
1662	6 months	11.76	2.14	12571	14472	342	2.31
2094	1 year	13.52	2.72	12220	14403	411	2.77
4101	3 years	14.26	2.29	11691	13890	924	6.23
7372	6 years	9.84	1.57	1202	13867	947	6.39
	STORED UNDER WATER						
1031	Same day	10.55	4.25	12891	14814	...	....
1081	Same day as submerged	13.98	2.65	12412	14716	98	0.66
1647	6 months	15.37	3.34	12013	14524	290	1.96
2100	1 year	13.85	3.81	12231	14517	297	2.00

NOTE: For Unit Coal formula, see footnote to Table 12.

Table 6
Williamson County Nut Coal

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
	STORED IN EXPOSED BINS						
1090 1091	Same day	13.98	3.73	12499	14859	...	....
1098	7 days	14.90	3.02	12341	14821	38	0.26
1246	2 months	14.32	4.12	12409	14835	24	0.16
1657	6 months	13.81	3.45	12455	14765	95	0.64
2090	1 year	11.88	2.73	12759	14734	125	0.84
4103	3 years	12.52	2.60	12503	14548	311	2.09
7374	6 years	12.86	2.20	12339	14406	453	3.04
	STORED IN COVERED BINS						
1090 1091	Same day	13.98	3.73	12499	14859	...	....
1098	7 days	14.90	3.02	12341	14821	38	0.26
1247	2 months	14.08	3.84	12378	14739	120	0.81
1663	6 months	13.06	3.60	12469	14644	215	1.45
2096	1 year	13.24	3.20	12428	14616	243	1.64
4100	3 years	13.93	2.93	12085	14323	536	3.60
7299	6 years	13.96	2.16	12103	14324	535	3.60
	STORED UNDER WATER						
1090 1091	Same day	13.98	3.73	12499	14859	...	....
1098	Same day as submerged	14.90	3.02	12341	14821	38	0.26
1648	6 months	15.65	3.12	12097	14673	186	1.25
2102	1 year	14.87	3.42	12251	14721	138	0.93

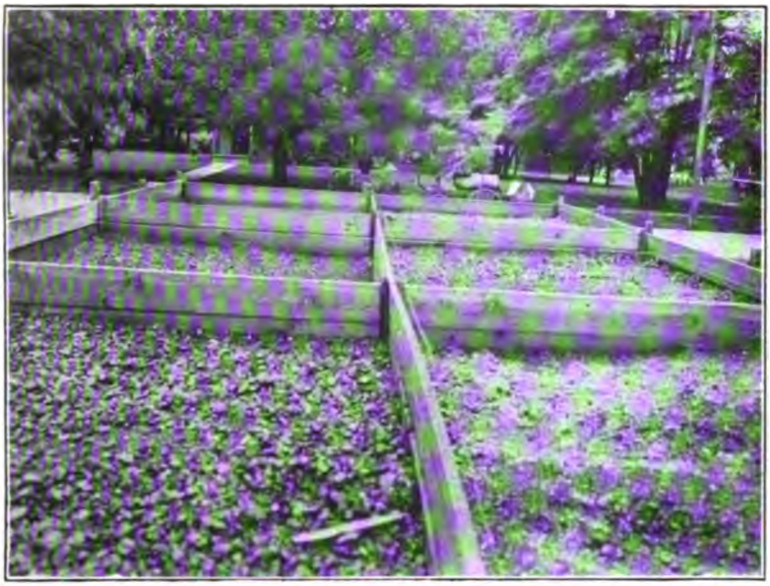

Fig. 1. Open Storage Bins

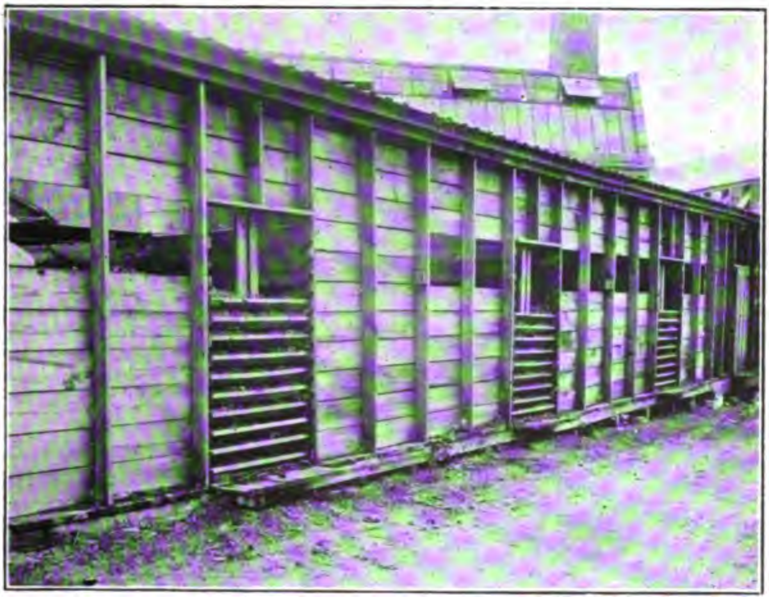

Fig. 2. Covered Storage Bins

TABLE 7

SANGAMON COUNTY NUT COAL

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
	STORED IN EXPOSED BINS						
1078	Same day	17.87	5.75	11741	14773	...	...
1084	7 days	16.63	5.10	11800	14571	202	1.37
1248	2 months	17.45	4.66	11626	14497	276	1.87
1658	6 months	16.03	4.91	11798	14444	329	2.23
2086	1 year	14.97	4.68	11860	14307	466	3.15
4107	3 years	15.55	4.13	11621	14102	671	4.54
7277	6 years	14.10	3.76	11180	13813	960	6.49
	STORED IN COVERED BINS						
1078	Same day	17.87	5.75	11741	14773	...	...
1084	7 days	16.63	5.10	11800	14571	202	1.37
1250A	2 months	16.08	5.03	11912	14600	173	1.17
1250B	2 months	17.57	5.01	11626	14535	238	1.61
1664	6 months	16.30	4.52	11682	14336	437	2.96
2092	1 year	15.99	4.65	11589	14165	608	4.12
4098	3 years	17.34	4.78	10681	13278	1495	10.11
7370	6 years	11.83	1.45	11971	13767	1006	6.81
	STORED UNDER WATER						
1078	Same day	17.87	5.75	11741	14773	...	...
1084	Same day as submerged	16.63	5.10	11800	14571	202	1.37
1649	6 months	15.90	4.21	11854	14461	322	2.18
2098	1 year	15.95	5.11	11851	14503	270	1.83

TABLE 8

VERMILION COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1032	Same day	17.88	2.35	11937	14888	...	...
1080	7 days	13.98	2.87	12414	14726	162	1.09
1082	7 days	13.69	2.29	12507	14759	129	0.87
1238	2 months	15.73	2.53	11958	14497	391	2.63
1239	2 months	14.69	2.90	12178	14578	310	2.08
1653	6 months	15.63	2.44	11969	14487	401	2.69
2089	1 year	14.46	2.24	12006	14304	584	3.92
4108	3 years	15.95	1.98	11229	13625	1263	8.48
7369	6 years	13.79	3.85	11210	13275	1613	10.83
STORED IN COVERED BINS							
1032	Same day	17.88	2.35	11937	14888	...	....
1080	7 days	13.98	2.87	12414	14726	162	1.09
1082	7 days	13.69	2.29	12507	14759	129	0.87
1241	2 months	15.26	2.51	12124	14608	280	1.88
1659	6 months	14.51	2.25	12071	14391	497	3.34
2095	1 year	15.36	2.42	11797	14225	663	4.46
4102	3 years	14.43	2.26	11199	13329	1559	10.47
STORED UNDER WATER							
1032	Same day	17.88	2.35	11937	14888	...	....
1080	Same day as submerged	13.98	2.87	12414	14726	162	1.09
1082	Same day as submerged	13.69	2.29	12507	14759	129	1.87
1644	6 months	13.87	2.32	12270	14514	374	2.51
2101	1 year	13.55	2.71	12283	14483	405	2.72

TABLE 9

WILLIAMSON COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
	STORED IN EXPOSED BINS						
1089	Same day.............	14.13	3.17	12426	14782	...	....
1099	7 days................	14.37	3.34	12287	14666	116	0.78
1244	2 months..............	15.66	2.67	12133	14701	81	0.55
1654	6 months.............	13.76	2.84	12342	14597	185	1.25
2091	1 year................	13.77	2.75	12328	14579	203	1.37
4106	3 years...............	13.25	2.26	13335	14470	312	2.11
7371	6 years...............	13.42	2.23	12077	14196	586	3.96
	STORED IN COVERED BINS						
1089	Same day.............	14.13	3.17	12426	14782	...	....
1099	7 days................	14.37	3.34	12287	14666	116	0.78
1245	2 months..............	12.62	2.98	12608	14705	77	0.52
1660	6 months.............	13.60	3.03	12372	14610	172	1.16
2097	1 year................	13.43	2.72	12385	14582	200	1.35
4099	3 years...............	13.07	2.66	12146	14230	552	3.73
7281	6 years...............	12.64	2.19	12153	14144	638	4.31
	STORED UNDER WATER						
1089	Same day.............	14.13	3.17	12426	14782	...	....
1099	Same day as submerged	14.37	3.34	12287	14666	116	0.78
1645	6 months.............	14.38	3.54	12262	14645	137	0.93
2103	1 year................	13.60	2.97	12447	14698	84	0.57

TABLE 10

SANGAMON COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
	STORED IN EXPOSED BINS						
1079	Same day	17.13	4.92	11752	14604	...	....
1085	7 days	17.04	4.47	11684	14481	123	0.84
1242	2 months	17.22	5.00	11645	14488	116	0.79
1655	6 months	17.02	4.54	11526	14281	323	2.21
2087	1 year	17.25	4.54	11153	13853	751	5.14
4104	3 years	15.94	4.03	11131	13565	1039	7.11
7373	6 years	14.95	3.32	11326	13604	1000	6.85
	STORED IN COVERED BINS						
1079	Same day	17.13	4.92	11752	14604	...	....
1085	7 days	17.04	4.47	11684	14481	123	0.84
1243	2 months	18.33	4.70	11414	14404	200	1.37
1661	6 months	17.30	4.67	11466	14263	341	2.33
2093	1 year	17.06	4.73	11248	13944	660	4.52
4097	3 years	18.71	4.86	10325	13072	1532	10.48
7279	6 years	17.39	3.81	10497	13025	1579	10.81
	STORED UNDER WATER						
1079	Same day	17.13	4.92	11752	14604	...	....
1085	Same day as submerged	17.04	4.47	11684	14481	123	0.84
1646	6 months	19.86	5.60	11127	14372	232	1.59
2099	1 year	18.27	4.81	11479	14478	126	0.86

6. *Indicated Heat Losses.*—An inspection of the tables shows that the heating value decreases most rapidly during the first week after mining and continues to decrease more and more slowly for an indefinite time. One per cent is about the average loss for the first week, and an additional loss of two or three per cent may occur by the end of the first year. At the end of the six-year period the indicated losses in some cases equal nearly eleven per cent.

These losses may be ascribed to three distinct causes:

(a) The escape of combustible gases.

(b) The absorption of oxygen.

(c) The increase in weight of the organic or combustible portion of the coal.

7. *Escape of Combustible Gases.*—From certain experiments given in a former bulletin* there is positive evidence that combustible gases exude from freshly mined coal. The extent of this loss, however, is very small. Porter and Ovitz† have carried out a quantitative measurement of such escaping gases and estimate in the case of a coal from Benton, Franklin County, Illinois, the total loss of heat values in a period of seventeen months to be 0.16 per cent. As this was an extreme case for this variety of coal it is evident that the heat loss due to the exudation of combustible gases is practically negligible.

8. *Absorption of Oxygen.*—In Bulletin 32‡ on the occluded gases in coal it is shown that freshly mined coal has a marked avidity for oxygen. It has been shown¶ that the union of oxygen with the coal is a chemical combination and not a simple absorption. While we should expect such chemical action to be accompanied by the generation of heat and consequently by a reduction in the heat value of the coal, there is little information available as to the extent of such loss.

Porter and Ralston§ show the positive formation of heat from oxygen combination but do not present any data as to the amount of this loss. Lamplaugh and Hill** have attempted to measure the amount of heat, and their results show an average for English coals of 3.3 calories for each cubic centimeter of oxygen absorbed. Winmill and Graham†† have carried the same line of experimentation further and have modified slightly the factor for the heat generated, their result showing 2.1 calories per cubic centimeter of oxygen absorbed. The difference is, for the purpose of the present discussion, not material since the desired result is the approximate heat loss due to the oxygen combinations at ordinary temperatures. The maximum absorption under the most favorable circumstances ranges from 7 to 8 cc. oxygen per gram of coal as recorded in the experiments by Lamplaugh and

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*Parr and Wheeler, "The Deterioration of Coal Samples," Univ. of Ill. Eng. Exp. Sta. Bul. 17, p. 33.

†U. S. Bureau of Mines, Technical Paper No. 2, 1910.

‡Parr and Barker, "The Occluded Gases in Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 32.

¶Parr and Hadley, "The Analysis of Coal with Phenol as a Solvent," Univ. of Ill. Eng. Exp. Sta. Bul. 76.

§U. S. Bureau of Mines Tech. Paper 65, p. 8, 1914.

**Trans. Inst. Mining Engrs., Vol. 45, p. 629, 1913.

††Winmill and Graham, "The Absorption of Oxygen by Coal," Colliery Guardian, Sept. 11-18, 1915.

Hill and also by Winmill and Graham. Porter and Ralston found in the case of a Franklin county, Illinois, coal exposed for five months at ordinary temperatures to pure oxygen, an absorption at the rate of 5.3 cc. per gram of coal. This is approximately 10 calories per kilo of coal or about 0.12 per cent of the heat value of the coal. The loss of heat thus represented is so small that it is negligible as a factor effecting deterioration in the quality of the coal. It may, however, possess significance in the matter of spontaneous heating. This phase of the subject will be hereinafter discussed.

9. *Increase in Weight.*—It is evident that, if in the processes which attend the weathering of coal there is an increase in weight of the coal substance, the indicated heat losses are more apparent than real. For example, if at the beginning of the storage period a pound of the unit coal substance shows a value of 14,700 B.t.u., and at the end of the period the original pound has increased in weight by absorption or additions, say 5 per cent, then the heat value per pound of the resulting material will be 14,000 B.t.u., thus indicating an apparent loss of 4.76 per cent. Evidence from many sources has accumulated to show that coal exposed to air or oxygen increases in weight. In the experiments described by Parr and Hadley* it is shown that under certain conditions in the residue insoluble in phenol, which is regarded as the degradation product of the cellulose constituent, there is as much as 3 per cent increase in weight due to the taking up of oxygen. This work further shows that such additional oxygen is chemically combined and not merely absorbed. Other experimenters have presented evidence to the effect that coal increases in weight. Somermeier† has shown an increase due to oxidation for an Illinois coal 2.47 per cent. Porter and Ralston ‡ show also a measurable increase in weight due to oxygen absorption. In their experiments on oxidation at various temperatures they note that Illinois and Pittsburgh coals ''show increases of weight up to 260 degrees in spite of the loss of carbon and hydrogen in CO_2, CO and H_2O.'' Study of some of the values presented in Table 11 seems to give

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*"The Analysis of Coal with Phenol as a Solvent," Univ. of Ill. Eng. Exp. Sta. Bul. 76.

†Professor Somermeier was doubtless the first to suggest that the indicated decrease in heat values was in reality the result of an increase in weight. N. W. Lord and E. E. Somermeier, U. S. Geol. Surv. Bul. 323, p. 22.

‡U. S. Bureau of Mines, Tech. Paper 65, p. 20-22, 1914.

further evidence of an increase in weight of the organic or combustible part of the coal. Some of these data with a discussion of their bearing on this point are presented under the following topic.

10. *Decrease of Ash Percentages.*—A study of the relative ash values corresponding to the various stages of indicated heat losses reveals a consistent lowering of the percentages of ash. This result obviously is normal since the actual increase in the organic constituents of coal during storage with the corresponding increase in the weight of any given sample must result in a relatively lower amount of ash, or an apparent decrease in ash.

Of course, the exact duplication of these theoretical conditions was impossible in these experiments. The oxidation of the sulphur varied and the leaching out of the soluble sulphates would alter the ash values in a corresponding degree. Other variables might enter, such as irregularities in sampling or the possible accumulation of foreign or earthy matter. Notwithstanding all these possibilities of inaccuracy there is a striking consistency of results with reference to the decrease of indicated ash percentages as the process of weathering proceeded. The ash values of the several masses taken at the beginning and at the close of the six-year period are presented in Table 11.

TABLE 11

INDICATED ASH AT THE BEGINNING AND AT THE END OF SIX YEARS IN STORAGE IN OPEN BINS

No.	Coal and County	Average Ash Values as Determined by Analysis of 3 Samples (1) at the Mine (2) at Unloading of Car (3) after 2 Months	Average Ash Values after 6 Years as Shown by Analysis of 2 Samples (1) from Open Bins (2) from Covered Bins	Indicated Decrease in Ash
1	Nut—Sangamon........	17.01	12.96	4.05
2	Nut—Vermilion........	12.95	11.45	1.50
3	Nut—Williamson.......	14.32	13.41	0.91
4	Screenings—Sangamon..	17.43	16.17	1.26
5	Screenings—Vermilion ..	15.20	13.79	1.41
6	Screenings—Williamson.	14.19	13.03	1.16

Inspection of the ash values presented in Tables 5 to 10, inclusive, further emphasizes the results shown by Table 11. The factors for the submerged coal have not been included in Table 11. Although these samples were under test for only a year, they show a constancy in the ash values and an absence of change, which is in keeping with the uniformity of heat values credited to the unit coal substance.

The increase in weight of the coal in storage by addition to the organic constituents is further illustrated by the values presented in Table 12. The values presented in Column 1 represent the heat value for the unit coal when fresh, as found by averaging the unit coal values for three samples taken at the mine, at the time of unloading, and after two months in storage. The calorific values of the dry coal in open storage at the end of six years are shown in Column 2. The accompanying ash and sulphur values are shown in Columns 3 and 4. Column 5 shows the percentages of ash or inert material which must be present in order that the fresh unit coal

TABLE 12

INCREASES IN WEIGHT OF COAL DURING STORAGE, IN RELATION TO THE INDICATED DECREASE IN HEAT VALUE

No.	Coal and County	B. t. u. per Lb. of Fresh Coal Average of 3 Samples Taken (1) at Mine (2) at Unloading of Car (3) after 2 Mos. Storage	B. t. u. per Lb. after 6 Years Open Storage, Dry Coal Basis	Ash as Weighed Dry Coal Basis, with Heat Values as in Col. 2	Sulphur in Dry Coal after 6 Years Open Storage	Ash Plus Additive Material Acquired in 6 Years Open Storage	Ash as in Col. 3 but Corrected for Sulphur as in Formula for Unit Coal	Showing Amount of Additive Material by Difference
		1	2	3	4	5	6	7
1	Nut— Sangamon.......	14,614	11,180	14.10	3.76	23.5	17.3	6.2
2	Nut— Vermilion........	14,700	11,984	13.06	1.58	18.5	14.9	3.5
3	Nut— Williamson.......	14,838	12,339	12.86	2.20	16.9	15.1	1.8
4	Screenings— Sangamon.......	14,524	11,326	14.95	3.32	21.9	17.9	4.0
5	Screenings— Vermilion........	14,717	11,210	13.79	3.85	23.8	17.0	5.2
6	Screenings— Williamson......	14,716	12,077	13.42	2.23	17.2	15.7	1.5

For discussion of Unit Coal and Corrected Ash see Univ. of Ill. Eng. Exp. Sta., Bul. 37, p. 33.

$$\text{Unit B. t. u.} = \frac{\text{Indicated (dry) B. t. u.} - 5000 \text{ sulphur}}{1.00 - (1.08) + 22/40 \text{ sulphur}}$$

Corrected (dry) ash = ash as weighed × 1.08 + 22/40 sulphur.

values of Column 1 may after six years drop to the values shown in 3. These percentages are derived by the formula,

$$100 - \frac{\text{B.t.u. per pound after storage}}{\text{B.t.u. per pound of fresh coal}}$$

The values in Column 6 are the corrected ash values for the coal after six years in open storage.*

The difference between the apparent corrected ash values of Column 6 and the required ash for producing the values shown in Column 2 is a measure of the additive material which is assumed to have been taken up by the organic constituents as absorbed oxygen or hydroxyl additions. These additions bear a certain general relation to the indicated decrease of ash shown in Table 11 and, of course, thus bear a more direct relation to the indicated deterioration percentages (see Tables 5 to 10).

These facts taken together, therefore, seem to afford added basis for the statement that the actual losses of heat values in stored coal are apparent rather than real, and that the true heat losses are those due to escaping combustible gases and to the heat generated by direct combination of oxygen both of which have been shown to be practically negligible.

IV. Moisture Values for Weathered Coal

The moisture values for the stored coal at the end of the six-year period are presented in Table 13. Some recent work on the Properties of the Water in Coal† by Porter and Ralston suggests a relation between the type of coal and the amount of "inherent" or hygroscopic moisture retained by the coal upon air-drying. It might be argued, therefore, that if the coals here considered during storage altered in type, possibly by a reversion toward the lignitic form, then a correspondingly high percentage of the moisture should be retained on air drying. The values shown in the table are without significance as far as this theory is concerned. However, the data should be presented for other reasons. For example, the high percentage of total moisture is in a general way characteristic of the several kinds of coal in that it is affected by the degree of subdivision or disintegration which has taken place. As may be seen by reference to the discussion concerning the slacking of these coals, the coals which have undergone the greatest disintegration have the highest percentage of total moisture present. This result, however, is to be expected and is due to physical rather than to chemical action.

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*Corrected Ash (dry) = ash × 1.08 + $\frac{22}{40}$ × sulphur (see formula for Unit Coal, footnote to Table 12).

†H. C. Porter and O. C. Ralston, U. S. Bureau of Mines, Technical Paper 113, 1916.

V. SUMMARY OF CHEMICAL STUDIES

The results presented in the foregoing tables and the data analyzed in the discussions may be summarized as follows:

(1) After a period of one year, the indicated loss of heat values is relatively low, averaging about 3 or 3½ per cent.

(2) Different coals vary in indicated heat losses, those from the southern Illinois districts showing less change than those from the central part of the state. Exposure beyond a period of one year accentuates this difference between coals from different localities. The denser coals, such as those from Williamson county, undergo but little additional change, while the coals from the northern parts of the state show a decrease in the indicated heat values. The difference, however, does not exceed about 10 per cent.

(3) Deterioration is consistently greater in the case of screenings than in that of screened nut.

(4) Since the heat values are referred to the unit coal basis, that is, the moisture, corrected ash, and sulphur free material, and since an actual loss of heat values by ordinary processes of oxidation would result in the formation of CO_2 and H_2O, both volatile under the conditions, it follows that the heat losses are largely relative, since they must result from a relative increase in weight of the organic substance of the coal. For example, any increase in weight, as by the addition of oxygen to the chemical structure of the organic material, would result in a lower indicated heat value per pound of the unit substance as compared with the heat value per pound before such addition had occurred. Similarly, changes in the sulphur combination due to oxidation would increase the weight of the

coal in a manner not taken into consideration in deriving the "corrected" ash values. Thus, where FeS_2 becomes $2FeSO_4 + 7H_2O$, the relative weights for sulphur are in the ratio of 1:7; that is, the content of sulphur compounds as the result of the new combinations has increased in weight seven times.

(5) Storage in *open bins* shows quite consistently a *lower* percentage of loss of heat value per pound than *is the case with storage under cover.* This is easily understood when it is considered that the additional material formed, due to the oxidation of the sulphur, is soluble and tends to leach out under the long continued application of water resulting from exposure to the elements. For this reason, therefore, the increase in weight is greater in the case of the coal under cover than in that of the coal exposed in open bins. The resulting unit coal values should, therefore, be higher for the coal stored in open bins, or the leached coal, than for the coal stored under cover or the unleached coal. Comparisons as to the relative values in covered and uncovered bins should, however, be made only up to and including the three-year period. The removal to new locations of the bins and piles shortly before the three-year old samples were taken resulted in the reconstruction of the covered bins with flat and leaky roofs through which the water had more or less free access to the coal.

(6) The extent of oxidation or increase in weight is a function of the character of the coal. The coal in which the cellulose residuum seems to predominate has the greater avidity for oxygen, and the coal in which the resinic residuum predominates is less affected. The tabulation of the samples, therefore, is in the order of such activity, namely, Sangamon, Vermilion, and Williamson counties. This feature is consistent with the studies in the absorptive capacities for oxygen of various coals as carried on by Porter and Ralston and is especially of interest in connection with the studies of Dr. Hadley on the relative avidity for oxygen of the cellulose residuum as compared with the absorptive capacity of the resinic bodies, separation into these two type components being affected by means of phenol.*

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*"The Analysis of Coal with Phenol as a Solvent," Univ. of Ill. Eng. Exp. Sta. Bul. 76, p. 21, 1914.

According to the results derived by Dr. Hadley, the cellulose residue had a far greater avidity for combination with oxygen than the resinic material.

TABLE 13

PERCENTAGE OF MOISTURE IN WEATHERED COAL, LOSS DUE TO AIR-DRYING, AND THE AMOUNT RETAINED BY THE AIR-DRIED SAMPLES

Table No.	Lab. No.	Coal, County and Condition	Total Moisture	Loss Due to Air-Drying	Moisture Retained in the Air-Dried Sample
1	7369	Screenings Vermilion Exposed Bin	17.70	14.15	4.14
2	7370	Nut Sangamon Covered Bin	19.95	16.54	4.09
3	7371	Screenings Williamson Exposed Bin	12.01	8.59	3.74
4	7372	Nut Vermilion Covered Bin	17.79	14.50	3.85
5	7373	Screenings Sangamon Exposed Bin	19.26	15.79	4.12
6	7374	Nut Williamson Exposed Bin	9.77	7.05	2.93
7	7277	Nut Sangamon Exposed Bin	20.35	17.01	4.03
8	7279	Screenings Sangamon Covered Bin	22.04	19.36	3.33
9	7298	Nut Vermilion Covered Bin	20.02	16.01	4.77
10	7299	Nut Williamson Covered Bin	12.54	9.45	3.30

VI. CHANGES IN PHYSICAL PROPERTIES

11. *Sizing Test.*—The extent of the disintegration or "slacking" which takes place in connection with the storing of coal is a matter of considerable importance because of the effect upon combustion on the grates where a large amount of finely divided material is present In order to determine the effect of storage upon slacking the sizing tests were continued to cover the entire period of six years. The

tests were made with a revolving screen with round perforations, as shown in Fig. 3.

Three sizing tests were made: one at the time the coal was placed in storage, one after a period of eighteen months, and one at the end of six years. It should be recalled that approximately three years before the last screening tests the storage piles were removed by wagon to a new location involving a double handling. Since the

TABLE 14

RESULTS OF SIZING TESTS ON NUT COAL STORED IN OPEN BINS

Round-Hole Screen		Original Sizes		In Storage for 1½ Years		In Storage for 6 Years	
Through Inches	Over Inches	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent
			SANGAMON COUNTY				
3	1	89.4	....	64.3	....	30.9	....
1	¾	4.1	93.5	6.9	71.2	9.6	40.5
¾	½	3.5	97.0	8.4	79.6	15.9	56.4
½	⅜	1.2	98.2	3.2	82.8	8.5	64.9
⅜	¼	0.6	98.8	4.0	86.8	3.2	68.1
¼	⅛	0.6	99.4	7.4	94.2	17.0	85.1
⅛	0	0.6	100.0	5.8	100.0	14.9	100.0
Total		100.0		100.0		100.0	
Average Diameter			1.854 inches		1.442 inches		.889 inches
			VERMILION COUNTY				
3	1	66.2	....	42.5	....	25.0	....
1	¾	5.0	71.2	8.0	50.5	6.3	31.3
¾	½	7.2	78.4	11.8	62.3	12.5	43.8
½	⅜	4.0	82.4	6.9	69.2	8.7	52.5
⅜	¼	4.0	86.4	6.8	76.0	12.5	65.0
¼	⅛	5.0	91.4	10.9	86.9	16.3	81.3
⅛	0	8.6	100.0	13.1	100.0	18.7	100.0
Total		100.0		100.0		100.0	
Average Diameter			1.458 inches		1.074 inches		.753 inches
			WILLIAMSON COUNTY				
3	1	94.0	....	70.2	....	60.6	....
1	¾	1.6	95.6	5.7	75.9	9.1	69.7
¾	½	1.8	97.4	6.6	82.5	8.3	78.0
½	⅜	0.7	98.1	3.1	85.6	2.7	80.7
⅜	¼	0.5	98.6	3.2	88.8	3.7	84.4
¼	⅛	0.5	99.1	4.6	93.4	5.5	89.9
⅛	0	0.9	100.0	6.6	100.0	10.1	100.0
Total		100.0		100.0		100.0	
Average Diameter			1.910 inches		1.532 inches		1.383 inches

extent of disintegration was not materially different for the coals stored in the exposed bins and for those stored in the covered bins, and since during the last years of storage the covered bins were almost as much exposed to rain and weather conditions as the uncovered bins, the results of sizing tests on coals stored in covered bins are omitted. Those for the coals stored in open bins are presented in Tables 14 and 15.

TABLE 15

RESULTS OF SIZING TESTS ON SCREENINGS IN OPEN BINS

Round-Hole Screen		Original Sizes		In Storage for 1½ Years		In Storage for 6 Years	
Through Inches	Over 1 Inch	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent
SANGAMON COUNTY							
1¾	1.0	38.8	....	15.1	....	10.0	....
1.0	¾	7.9	46.7	9.3	24.4	8.2	18.2
¾	½	13.2	59.9	15.6	40.0	14.5	38.7
½	⅜	6.6	66.5	7.7	47.7	10.9	43.6
⅜	¼	7.2	73.7	9.4	57.1	15.4	59.0
¼	⅛	11.2	84.9	17.2	74.3	24.5	83.5
⅛	0	15.1	100.0	25.7	100.0	16.5	100.0
Total..............		100.0		100.0		100.0	
Average Diameter ..			.768 inches		.498 inches		.452 inches
VERMILION COUNTY							
1¾	1	19.0	....	11.3	....	8.8	....
1.0	¾	8.9	27.9	6.3	17.6	6.3	15.1
¾	½	14.8	42.7	12.9	30.5	12.1	27.2
½	⅜	8.5	51.2	9.3	39.8	9.3	36.5
⅜	¼	11.1	62.3	11.8	51.6	13.9	50.4
¼	⅛	16.4	78.7	21.0	72.6	29.0	79.4
⅛	0	21.3	100.0	27.4	100.0	20.6	100.0
Total..............		100.0		100.0		100.0	
Average Diameter ..			.548 inches		.425 inches		.403 inches
WILLIAMSON COUNTY							
1¾	1	18.9	....	19.0	....	6.0	....
1.0	¾	9.0	27.9	9.2	28.2	5.5	18.2
¾	½	14.4	42.3	15.4	43.6	11.5	32.7
½	⅜	8.5	50.8	9.5	53.1	9.3	43.6
⅜	¼	10.4	61.2	10.6	63.7	14.4	59.0
¼	⅛	15.4	76.6	17.0	80.7	32.2	83.5
⅛	0	23.4	100.0	19.3	100.0	21.1	100.0
Total..............		100.0		100.0		100.0	
Average Diameter ..			.542 inches		.557 inches		.255 inches

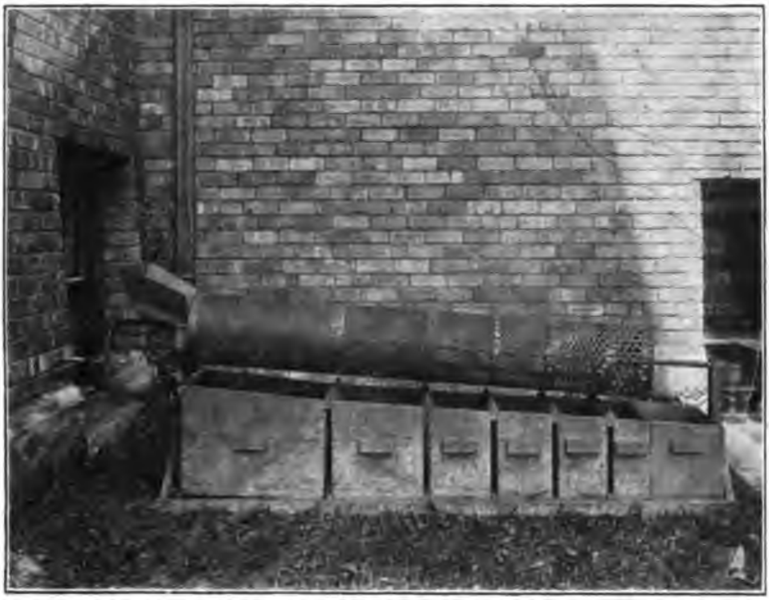

Fig. 3. Revolving Screens used in Sizing Tests
(Reproduced from Univ. of Ill. Eng. Exp. Sta. Bul. 78, 1909)

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A condensed summary of the detailed results presented in Tables 14 and 15 is given in Table 16. The total dust passing through a ¼-inch screen is taken as the factor indicating the increase of fine material, and percentages of increase for the two periods are based on the total amount of material in the original coal over ¼-inch in size. Moreover, the samples taken from this table vary with reference to the method of storage, three being from exposed and three from covered bins. These particular samples are used in the table for the reason that the same storage lots were selected for making the standard boiler tests which are hereinafter discussed.

TABLE 16

INCREASE IN FINE MATERIAL AFTER ONE AND ONE-HALF AND SIX YEARS (BASIS OF REFERENCE, THE TOTAL COARSE MATERIAL IN THE ORIGINAL COAL PASSING OVER ¼-INCH SCREEN)

Table No.	Coal and County	How Stored	Initial Storage	After 1½ Years		After 6 Years	
			Dust Passing ¼-Inch Screen	Dust Passing ¼-Inch Screen	Percentage Increase of Fine Material Referred to Original Coal over ¼-Inch	Dust Passing ¼-Inch Screen	Percentage Increase of Fine Material Referred to Original Coal over ¼-Inch
1	Nut—Sangamon.....	Open	1.2	13.2	12.1	31.9	31.0
2	Nut—Vermilion......	Open	13.6	24.0	12.0	35.0	24.7
3	Nut—Williamson....	Covered	1.4	11.0	11.1	13.9	12.6
4	Screenings—Sangamon.....	Covered	26.3	38.5	16.5	45.1	25.5
5	Screenings—Vermilion......	Open	37.7	48.4	17.1	49.6	19.1
6	Screenings—Williamson....	Covered	38.8	45.4	10.7	50.6	19.2

An examination of Table 16 shows that the rate of disintegration is consistent with the variety of coal as already discussed in connection with the absorptive capacity for oxygen, and suggests that the process of oxidation of the organic material may be quite as largely responsible for this breaking down of the particles as the oxidation of the finely divided pyrite sulphur. The latter may or may not be distributed throughout the texture of the coal; this characteristic, for Illinois coal at least, has not been determined.

VII. BOILER TESTS ON WEATHERED COAL

Upon the completion of the storage experiments at the end of the six-year period, it was decided to conduct if possible a series of boiler tests under standard conditions and to compare the results of tests with stored coal with those of similar tests in which fresh coal is used. Such an opportunity presented itself in connection with a series of boiler tests being conducted at that time by Mr. A. P. Kratz, who was making a general study of boiler losses.*

The coal which Mr. Kratz was using as standard material consisted of screenings from the Mission Field, Vermilion county, Illinois. Nineteen tests were conducted with this coal. Five additional tests were made on samples of the weathered coal, one sample each of nut coal being selected from the lots from Sangamon, Vermilion and Williamson counties, and one sample each of screenings from the Sangamon and the Williamson county lots. With reference to the ash content and the percentage of finely divided material, the properties of the stored coals did not differ greatly from those of the fresh coals. An excellent basis was, therefore, provided for comparing the efficiency of the stored and fresh coals. As might be expected, some experience had to be acquired as to the best method of handling and firing the weathered coals, and although the first test was unsatisfactory, subsequent experiments yielded results which proved trustworthy and useful as a basis of comparison.

"On the first test the coal banked slightly at the water-back, and the whole amount on the grate became clinkered. It immediately became evident that in order to run at all, the coal had to be kept away from the water-back. After the clinker had been removed a fresh start was made, and care was taken to keep the fuel bed from four to six inches away from the water-back. When this was done no further trouble was experienced."†

With reference to the relative drafts required for a given rate of combustion with the fresh and the weathered coal, the tests showed that a stronger draft is required for the weathered coal. Moreover, by comparing draft requirements for those lots which were in close agreement as to their dust content, it seems evident that the higher draft requirement is not necessarily due to a higher dust factor. The explanation for this is simple if we recall the discussion already presented concerning the avidity for oxygen of freshly mined coal. This avidity for oxygen seems to be directly related to the free burning character of the coal. Pillar coal and coal that has been long in storage does not burn so freely as fresh coal. On account of this characteristic such coal is thought to have lost a large part of its heat, when as a matter of fact it may have the same number of heat units but a very different rate of combustion. Another factor, and probably a minor one, is the loss of combustible gases. Mention has been made of the fact that part of the deterioration which occurs in the first few days or weeks after mining is due to the escape of certain light volatile or gaseous fuel constituents. It is evident, therefore, that either on account of the avidity of the fresh unsaturated coal for oxygen or the presence of light fuel constituent, or because of both conditions combined, the fresh unweathered coal burns freely and gives up its heat units readily, whereas the opposite is true with weathered coal. This difference, therefore, must be offset by a stronger draft or by some combination of conditions which will effect a speeding up of the burning or oxidation process. If by this means the rate of combustion can be made to approach that of the fresh coal, a corresponding degree of efficiency should result. The correctness of this theory is borne out by the results shown in Table 17. It is to be noted that the over-all efficiency of

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*A. P. Kratz, "Study of Boiler Losses," Univ. of Ill. Eng. Exp. Sta. Bul. 78, 1915.

†Univ. of Ill. Eng. Exp. Sta. Bul. 78, p. 50, 1915.

TABLE 17

RESULTS OF BOILER TESTS WITH MISSION FIELD FRESH COAL AND WITH WEATHERED COAL AFTER SIX YEARS IN STORAGE

Mission Field			Weathered Coal				
Test No.	Boiler H. P. Developed	Efficiency of Boiler, Furnace, and Grate Per Cent	Test No.	No.	Coal and County (Tables 5 to 10)	Boiler H. P. Developed	Efficiency of Boiler, Furnace, and Grate Per Cent
10	554.0	63.96	20	1	Nut—Sangamon.......	568.5	64.50
11	569.6	61.21	21	4	Screenings—Sangamon.......	557.2	63.05
12	572.7	60.67	22	3	Nut—Williamson......	727.1	65.98
13	589.0	69.87	23	6	Screenings—Williamson......	509.6	60.04
14	555.9	64.75	24	2	Nut—Vermilion........	655.0	64.20
15	506.6	65.50					
16	644.0	60.84					

the weathered coal averages quite as high as that of the fresh screenings.

The general summary covering the behavior of the coal in steam generation after six years of storage, as set forth in Bulletin 78 of the University of Illinois Engineering Experiment Station, is as follows:

> "1. Burning weathered coal is largely a question of correct handling and ignition. Under these circumstances it gives as good results as fresh screenings.
>
> "2. Weathered coal requires a little thinner fire and more draft than fresh screenings.
>
> "3. When using weathered coal the fuel bed should not approach any nearer to the water-back than from 4 to 6 in., otherwise trouble with clinker is experienced.
>
> "4. Practically as high capacity was obtained with weathered coal as with the other coals used, and, if anything, the fuel bed requires less attention."

In this reference, attention is called further to the fact that the results obtained and the conclusions presented are based on the heat values in the coal as fired and do not take into account the matter of deterioration. But it has already been shown in the previous discussion that the deterioration is largely apparent in a physical change and that the actual loss of heat value is small. Hence, the efficiency factors developed in the tests may be accepted as fairly representing results obtainable on weathered coal in which the heat loss resulting from weathering is practically negligible.

VIII. CONCLUSIONS

The facts presented in the preceding sections of this bulletin justify the following conclusions:

(1) Bituminous coal can be stocked without appreciable loss of heat values provided the temperature is not allowed to rise above 180 degrees F. In fact, there is no appreciable evolution of CO_2 at temperatures below 260 degrees F.

(2) The indicated heat loss per pound of coal is due more largely to an increase in weight of a unit mass of coal resulting from the absorption of oxygen than to an actual deterioration or loss of heat units.

(3) Freshly mined coal has a large capacity for absorbing oxygen which combines chemically with both the organic combustible and the iron pyrites present.

(4) The combination of oxygen with coal at ordinary temperatures generates a small increment of heat.

(5) The rapidity with which oxygen is absorbed depends upon the temperature of the mass and the extent of the superficial area exposed, that is, the fineness of division of the coal.

(6) If heat is generated by this slow process of oxidation more rapidly than it is lost by radiation, the acceleration of the reaction causes a rise in temperature which quickly brings the mass up to the danger point. A temperature of 180 degrees F. is named as the danger point because, if the coal reaches that temperature, practically all of the free moisture is vaporized and the further rise in temperature will be very rapid.

(7) Any method of storage to be successful must either check or prevent the absorption of oxygen to such an extent that the generation of heat shall not proceed so rapidly as to exceed natural heat losses due to radiation.

(8) Underwater storage prevents loss of heat values, and is not accompanied by deterioration in physical properties, such as slacking.

(9) Dry storage is far more safely undertaken if the fine material is screened out at the storage yard and the lump only, preferably sized, is stocked.

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PUBLICATIONS OF THE ENGINEERING EXPERIMENT STATION

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THE COLLAPSE OF SHORT THIN TUBES

BY

A. P. CARMAN

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BULLETIN No. 99 JUNE, 1917

THE COLLAPSE OF SHORT THIN TUBES

BY
A. P. CARMAN
PROFESSOR OF PHYSICS

ENGINEERING EXPERIMENT STATION
PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

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THE COLLAPSE OF SHORT THIN TUBES

I. RESULTS OF PREVIOUS EXPERIMENTS ON COLLAPSE OF TUBES

1. *Fairbairn's Formula.*—The problem involved in the study of the collapse of tubes is to find an equation by the application of which the collapsing pressure of a given tube can be calculated when the dimensions of the tube and the elastic properties of the material are known. The first person to attempt a solution of the problem was the noted British engineer, Sir William Fairbairn. In 1858, Fairbairn published in the Philosophical Transactions of the Royal Society of London a paper describing experiments on the collapse of about twenty-five tubes. Fairbairn's work was done at the suggestion and with the aid of the Royal Society, and the British Association for the Advancement of Science. The immediate cause was the need of such knowledge in the design of steam boilers. The tubes were "composed of a single thin iron plate bent to the required form upon a mandril and riveted and also brazed to prevent leakage into the interior." The ends were closed by means of iron disks or plugs, and the tube was placed in a large iron cylinder where it was subjected to hydraulic pressure. The interior of the tested tube was connected with the atmosphere at the upper end. The pressure was produced by a hydraulic pump and was measured by ordinary steam gauges. The diameters of the tubes collapsed were 4, 6, 8, 9, 10, and 12 inches; the lengths ranged from 19 to 60 inches; and the wall thickness was .043 of an inch for each tube. The tubes were what we describe to-day as "short" and "thin."

Fairbairn summed up the results of his experiments in the well-known formula:

$$P = 9{,}675{,}600\,\frac{t^{2.19}}{l\,d} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

"which is," he says, "the general formula for calculating the strength of wrought iron tubes subjected to external pressure within the limits indicated by the experiments, that is, provided their length is not less than 1.5 feet and not greater than 10 feet." In this formula, as in other formulas presented in this bulletin, P is collapsing pressure in pounds per square inch; t is thickness of wall in inches; l is length of tube in inches; d is diameter in inches. It is noteworthy in the history of the subject that Fairbairn's formula in some form was the only available formula for engineers for nearly fifty years, and that his experiments seem to have been the only experiments made in this field prior to 1905 and 1906. Meanwhile, the materials and methods of construction, and the uses of tubes had changed so that Fairbairn's constants were wholly out of date.

In addition, Fairbairn's tubes, as has since been shown, were below a certain "critical length"; and, the fact of a minimum critical length not being known, the formula was generally assumed to be applicable to tubes of all lengths. Various modifications were proposed for the formula* but practically all of them were based on Fairbairn's data.

2. *Experiments of Carman.*—The first experiments on the collapse of tubes after Fairbairn's were those of A. P. Carman, which were reported and discussed in a paper read before the American Physical Society in April, 1905, and published in the Physical Review, Vol. XXI. These experiments were conducted with small seamless brass tubes and showed "that there is a minimum length for each tube above which the collapsing pressure is constant. Again," still quoting this paper, "we see that for lengths less than this critical minimum length, the collapsing pressure rises rapidly. As definitely as can be determined from these small tubes, the collapsing pressure varies inversely as the length, for lengths less than the critical length." It was further pointed out that Fairbairn's tubes were all less than the critical length. "These experiments on small brass tubes have shown that formulas of the Fairbairn type are inadequate, and that for tubes of sufficient length, a formula of the type proposed by Bryan and Love is more nearly true." The formula referred to is that first given by Professor G. H. Bryan† of Cambridge, England, and later deduced by other methods by A. B. Bassett‡ and A. E. H. Love§. The formula is given for long thin tubes, and is

$$P = 2E \frac{m^2}{m^2-1} \frac{t^3}{d^3} \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

where P is the pressure of collapse, t is the thickness of wall, d is the mean diameter, E is Young's modulus, and $\frac{l}{m}$ is Poisson's ratio.

---

*See Bibliography.

†Proceedings Cambridge Philosophical Society, Vol. VI.

‡Philosophical Magazine, September, 1892.

§Mathematical Theory of Elasticity, Vol. 2, p. 319, 1903.

3. *Results Obtained by Stewart, and by Carman and Carr.*—In 1906, two papers appeared on the collapse of tubes; one by Professor R. W. Stewart* read before the American Society of Mechanical Engineers, giving the results of a large number of experiments on lap-welded steel tubes; and the other by Carman and Carr† in a bulletin of the Engineering Experiment Station of the University of Illinois, describing experiments on seamless steel, lap-welded steel, and brass tubes. All of these experiments were made on tubes longer than the "critical length," little attention being paid to the law of lengths except to see that the tubes were longer than the critical minimum length.

The results obtained by Carman and Carr were stated as follows:

"The portion of a long tube affected by the collapse under hydraulic pressure is generally not greater than twelve times the diameter; for greater lengths the collapsing pressure is independent of the length. The law according to which the collapsing pressure varies inversely as the length, is true only for very short tubes; i. e., tubes shorter than a certain critical 'minimum length' which in most cases is from four to six times the diameter.

" For long thin tubes, i. e., for values of $\frac{t}{d}$ below about .025, the formula $P = k\,(\frac{t}{d})^3$ is very nearly true." The constants for brass and seamless tubes were calculated from the experimental data, and the formulas stated as follows:

a. For thin brass tubes

$$P = 25{,}150{,}000\,(\frac{t}{d})^3 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

b. For thin seamless steel tubes

$$P = 50{,}200{,}000\,(\frac{t}{d})^3 \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

" For moderately 'thick' tubes, that is, for tubes having values of $\frac{t}{d}$ between .03 and .07, an equation of the form

$$P = k\,\frac{t}{d} - c \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

---

*Transactions of the American Society of Mechanical Engineers, 1906.

†" Resistance of Tubes to Collapse." Univ. of Ill. Eng. Exp. Sta. Bul. 5, 1906.

was found to satisfy the experimental data, the formula becoming:

a. For brass

$$P = 93{,}365\frac{t}{d} - 2474 \quad . \quad . \quad . \quad . \quad . \quad . \quad (6)$$

b. For seamless cold drawn steel

$$P = 95{,}520\frac{t}{d} - 2090 \quad . \quad . \quad . \quad . \quad . \quad . \quad (7)$$

c. For lap-welded steel

$$P = 83{,}290\frac{t}{d} - 1025 \quad . \quad . \quad . \quad . \quad . \quad . \quad (8)\text{''}$$

Professor Stewart found similar equations and constants from his experiments on lap-welded steel tubes.

Thus from theory and from experiments, fairly satisfactory equations for long thin tubes and for long tubes of moderate thickness have been reached. The terms "long," "thin," and moderately "thick" are used as previously defined. The equation for long, moderately thick tubes is empirical both in its constants and its form. The equation for long thin tubes is that of Bryan except that the constant is about 30 per cent less than the theoretical value, probably due, as suggested by R. V. Southwell,* to the material being beyond the elastic limits.

4. *Investigations of Southwell, and of Cook.*—Within the last three years, there has been considerable interest in the law of the collapse of "short" thin tubes, that is, in the form of the pressure-length curve near and inside the critical length. The practical interest in this part of the curve came first from the problem of spacing "collapse rings" in boiler flues. Another practical interest in the problem is found in the collapse of steel flumes by atmospheric pressure when the water is suddenly let out by accident and the internal pressure is reduced almost to zero.

The theoretical interest comes from the very important mathematical investigations of R. V. Southwell, Fellow of Trinity College, Cambridge, England, discussed in a paper entitled "On the General Theory of Elastic Stability." This paper was read before

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*Philosophical Magazine, p. 504, September, 1913.

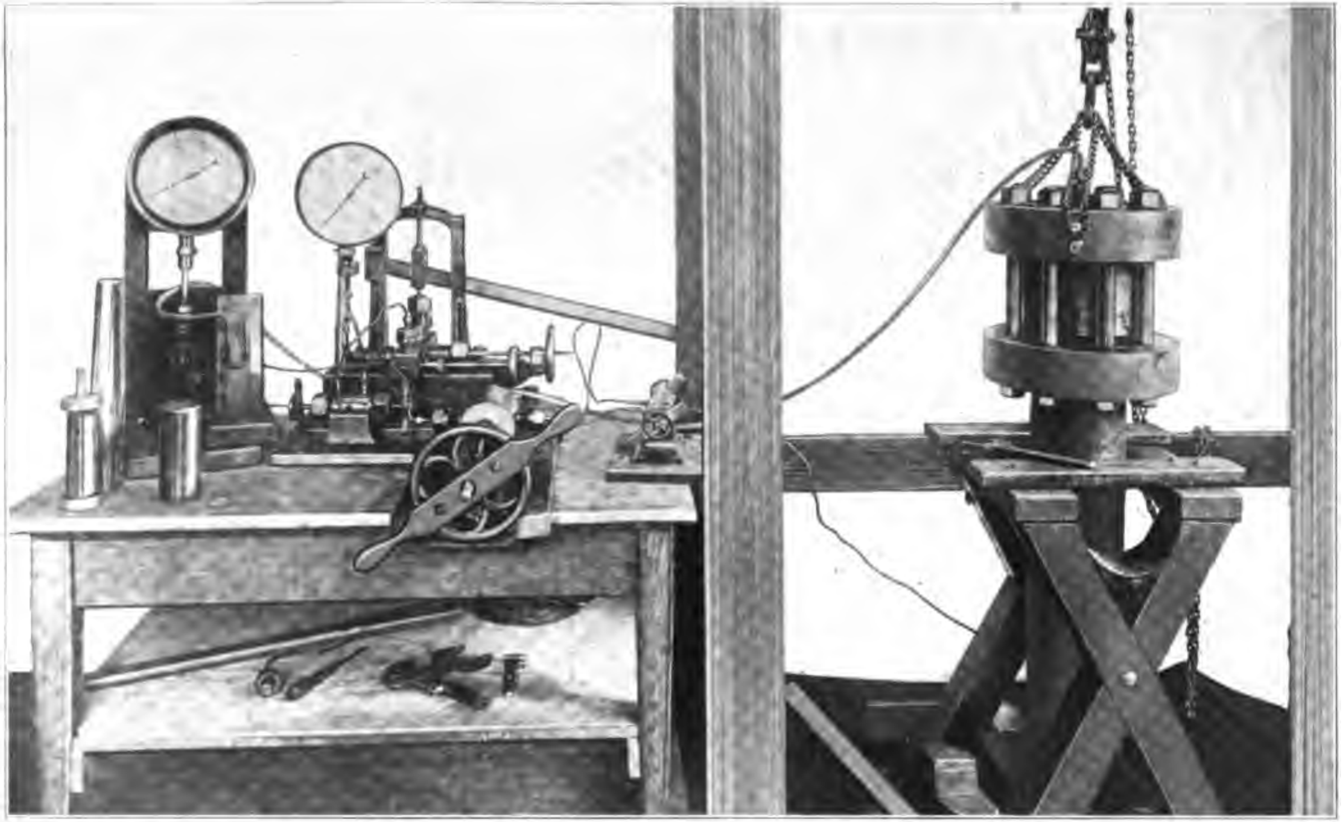

FIG. 1. APPARATUS ARRANGEMENT. PUMP AND GAUGES TO LEFT, NICKEL-STEEL CYLINDER TO RIGHT

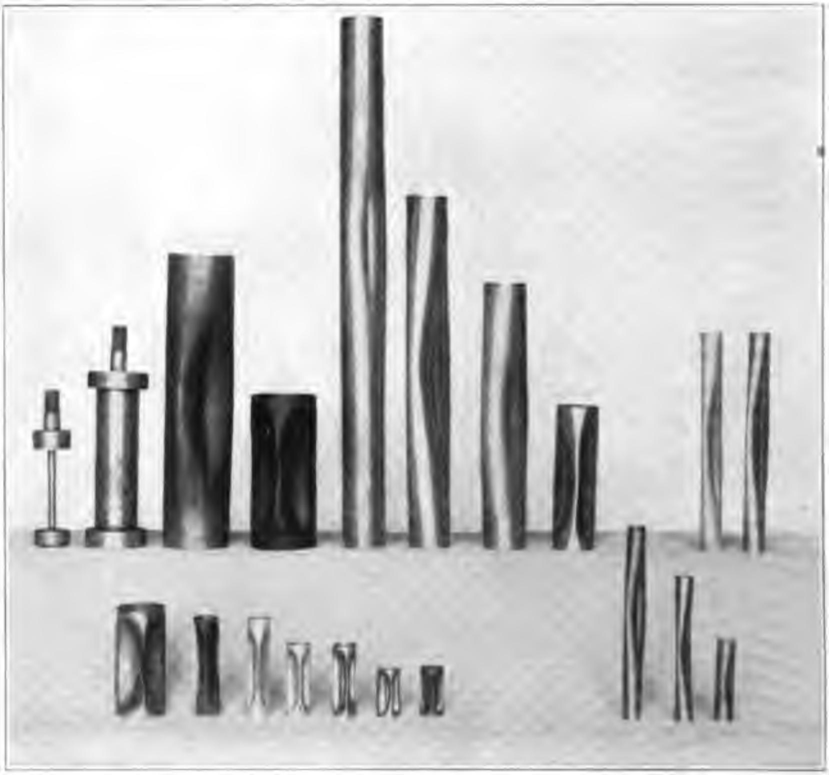

FIG. 2. TUBES THAT HAVE BEEN COLLAPSED. ON THE LEFT THE MOUNTING ARRANGEMENT IS SHOWN. LONG TUBES COLLAPSE IN TWO LOBES, SHORTER TUBES INTO THREE LOBES, AND STILL SHORTER TUBES INTO FOUR LOBES

the Royal Society of London in January, 1913.* In it Southwell developed the following formula for the collapse of thin tubes:

$$P = 2E\frac{t}{d}\left[\frac{Z}{k^4\,(k^2-1)}\,\frac{d^4}{l^4} + \frac{1}{3}\,\frac{m^2}{m^2-1}\,(k^2-1)\,\frac{t^2}{d^2}\right] \quad . \qquad (9)$$

where P is the collapsing pressure, E is Young's modulus, $\frac{l}{m}$ is Poisson's ratio, Z is a constant depending upon the end constraints, l, d, and t are the length, diameter, and wall thickness, and k represents the number of lobes into which the tube collapses. It is known that long tubes collapse into two lobes, shorter tubes into three lobes, and still shorter into four lobes (See Fig. 2). This formula is represented by a family of curves, corresponding to the value of 2, 3, and 4, for k. Fig. 3 is reproduced from Southwell's paper. It shows the curves

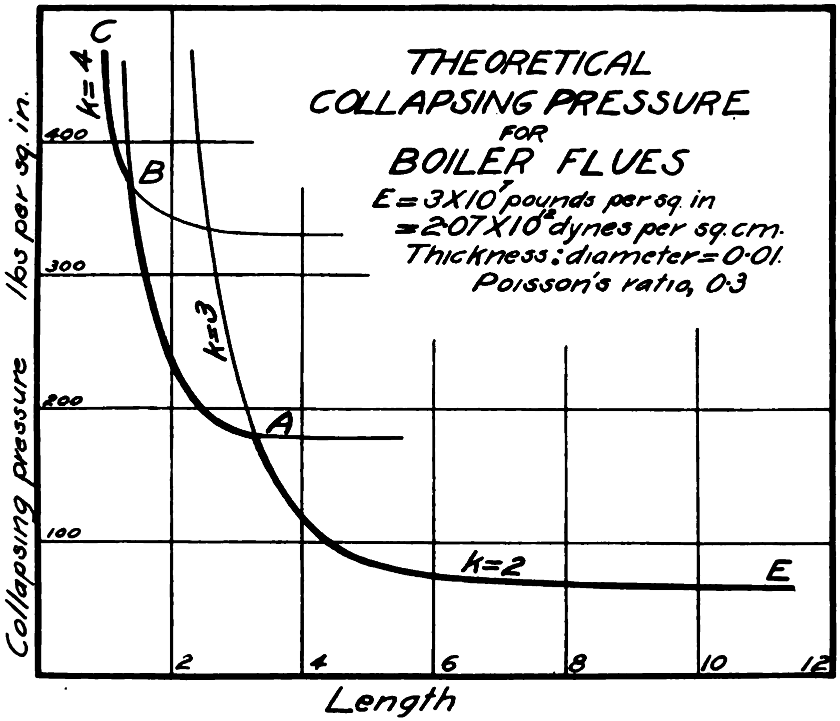

FIG. 3. CURVE OF SOUTHWELL'S EQUATION REPRODUCED FROM PHILOSOPHICAL TRANSACTIONS, 1913

for two, three, and four lobes, giving the relation between collapsing pressure and length; values of Young's modulus and Poisson's ratio are assumed as noted on the curve. No simple value for Z is stated by Southwell. "From an inspection of the different curves, we see," writes Southwell, "that long tubes will always tend to collapse into the two-lobed form, since the curve for $k = 2$ then gives the least value for the collapsing pressure, but that at a length corresponding to the point A, the three-lobed distortion becomes natural to the tube; and that for shorter lengths still, of which the point B gives the upper limit, the four-lobed form requires least pressure for its maintenance. Thus the true curve connecting pressure and length is the discontinuous curve *CBAE*, shown in the diagram by a thickened line." Then according to Southwell's theory, the curves which are ordinarily drawn to represent the results of experiments, such as the curves in Figs. 8 to 14, are really the envelopes of series of curves. Southwell made some experiments which seem to confirm his idea of the pressure-length curve. Our own experiments have not given any decisive proof of this feature of Southwell's formula. It is to be noted that for long tubes, that is, where $k=2$, Southwell's formula reduces to the Bryan formula.

---

*Philosophical Transactions of the Royal Society of London, Vol. 213A, pp. 187-244.

Southwell has further discussed the theory of the collapse of tubes in three articles in the Philosophical Magazine for 1913 and 1915.* The third of these articles is largely a discussion of his theory with reference to some very interesting experiments on the collapse of short tubes by Gilbert Cook of the University of Manchester.†

Southwell's formula differs from the previous formula of Bryan in giving an expression for the collapsing pressures for lengths less than the "critical length." He deduces as an expression for the "critical length," $L = k\sqrt{\frac{d^3}{t}}$, "$k$ being some constant depending upon the type of the end constraints."‡

The Southwell formula further makes the envelope of the pressure-length curve a hyperbola for lengths less than the critical length. Hence, if the critical length can be determined, the curve can be drawn and the strength of a short tube calculated. The experiments of Carman and of Stewart indicated a length of six diameters as the critical length.

---

*Philosophical Magazine, pp. 687-698, May, 1913; pp. 502-511, September, 1913; pp. 67-77, January, 1915.

†Philosophical Magazine, pp. 51-56, July, 1914.

‡Philosophical Transactions, Vol. 213A, p. 227.

Cook's experiments were made to determine the strength of short tubes and, as a part of the investigation, to determine the "critical length." He gives the results of the collapses of thirty-nine "solid drawn" steel tubes, all of three inches internal diameter, of values of $\frac{t}{d}$ varying between .0097 and .0206, and of lengths from 2.08 inches to 12.71 inches. Unfortunately Cook's apparatus limited him to tubes of less than thirteen inches in length, that is, to lengths of about four diameters, so that his curves do not reach the important bends or "critical" points. He, however, concludes from estimates that "the critical length varies from thirteen to eighteen times the diameter," and that it is given by Southwell's formula $L = k\sqrt{\frac{d^3}{t}}$, with $k = 1.73$ for his steel tubes. Southwell in his discussion of Cook's work and the "critical length" says,* "Both theory and experiment suggest that the length of a tube sensibly affects its resistance to external pressure only in the case of comparatively short tubes, and the earliest definitions of the term 'critical length,' given almost simultaneously by Professor A. E. H. Love as 'the least length for which collapse is possible under the critical pressure,' and by A. P. Carman as a 'minimum length, beyond which the resistance of a tube to collapse is independent of the length,' were in recognition of this fact. Professor Carman concluded further from the early experiments of Fairbairn and from others which he himself conducted, that 'the collapsing pressure varies inversely as the length, for lengths less than the critical length.' That is to say, the curve suggested by him as expressing the experimental relation between collapsing pressure and length for a tube of given thickness and diameter consists of two discontinuous branches; a straight line, representing constant collapsing pressure for all lengths above the critical length, and a rectangular hyperbola intersecting this line at a point corresponding to the critical length.

"If these views were adopted, the critical length for any definite size of tube may be determined from experiments by estimating: (1) the straight line, parallel to the axis of length, which best represents the collapsing pressure for tubes of considerable length, and (2) the hyperbola which agrees best with the results for the shorter tubes; their point of intersection gives the required value. This is substantially the procedure adopted by Mr. Cook, who finds that within the range of his experiments the critical length L, thus defined, is given satisfactorily by the formula.

---

*Philosophical Magazine, p. 68, January, 1915.

$$L = 1.73\sqrt{\frac{d^3}{t}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \qquad (10)$$

"Two of the five curves upon which Cook bases his conclusions are

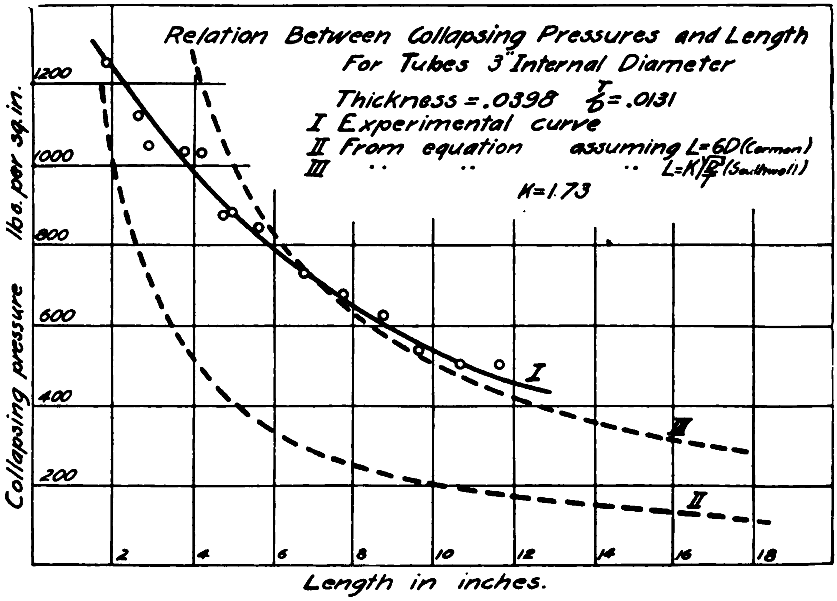

FIG. 4. PRESSURE-LENGTH CURVES SHOWING EXPERIMENTAL AND CALCULATED VALUES ACCORDING TO GILBERT COOK. REPRODUCED FROM PHILOSOPHICAL MAGAZINE, 1914

shown in Figs. 4 and 5 which are reproduced from his article. Agreement of the curve with this theory is not close, particularly for the thicker tubes. Although Cook suggests the given formula, he frankly says that 'the tests cannot be regarded as sufficient in number or covering enough range of dimensions to confirm definitely the equation $L = 1.73\sqrt{\frac{d^3}{t}}$ Though there is considerable amount of data for a law on long thin tubes and for long moderately thick tubes, there are few experimental results on the collapse of short tubes. The following experiments were undertaken in order to get more facts on the strength of short tubes of steel and of brass. To these have also been added some experiments on tubes of aluminum, glass, and hard rubber. The results obtained with the three last named materials not only have an interest in themselves, but also may have a theoretical interest in future discussions of formulas on account of the different elastic constants involved.

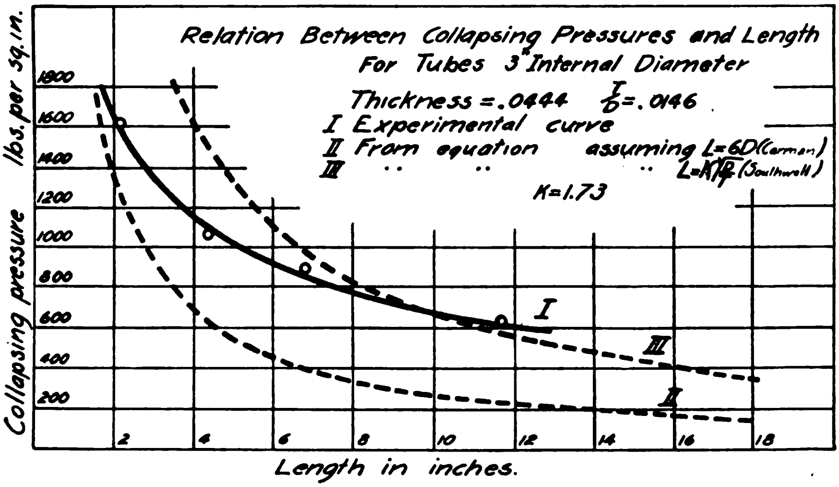

FIG. 5. PRESSURE-LENGTH CURVES SHOWING EXPERIMENTAL AND CALCULATED VALUES ACCORDING TO GILBERT COOK. REPRODUCED FROM PHILOSOPHICAL MAGAZINE, 1914

A large part of the experimental work described in this paper has been done by Stetfan F. Tanabe, M. S., Research Fellow of the Engineering Experiment Station. Credit is due him also for many of the details of the apparatus and methods.

II. METHODS AND RESULTS OF EXPERIMENTS ON THE COLLAPSE OF SHORT THIN TUBES

5. *Apparatus.*—The apparatus was for the most part that which the author used a number of years ago in previous experiments of this kind. The hydraulic pressure was produced in a stout nickel-steel tube or receptacle 40 inches in length, 5 inches in internal diam-

eter, and 7 inches in external diameter. The lower end was closed by a cast-iron plug, and the upper end by a heavy steel disk or cover which was held in place by eight circular bolts as shown in Figs. 5 and 6. A lead gasket with circular grooves in the end face of the tube prevented leakage even at the highest pressures. This tube was held in a vertical position, so that the heavy steel cover could be conveniently

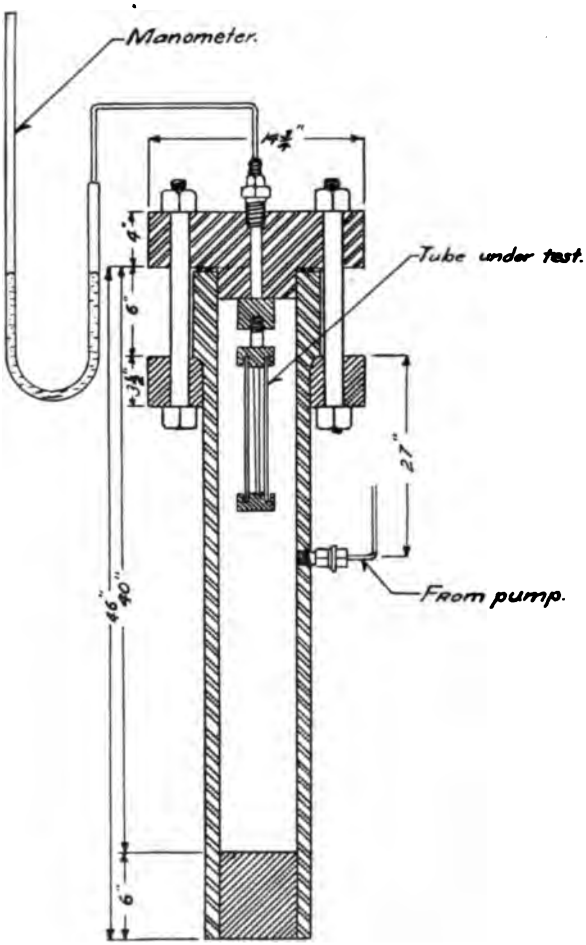

Fig. 6. Section of the Nickel-Steel Tube in which the Tubes were Collapsed by Hydraulic Pressure

handled by a chain hoist. The hydraulic pressure was produced by a Cailletet pump, made by the Société Génevoise, and by a screw plunger, which was made in the department shop. The pressure was raised nearly to the required amount by the pump, and then brought gradually to the pressure of collapse by means of the screw plunger. It was measured by three tested gauges of the Bourdon type. The arrangement for closing the tube for collapse is shown in Fig. 7. This was the result of several trials, and proved very effective and easy to use. The steel end plugs were turned so that the tube could slide over them with a close fit, but without strain. The length of the

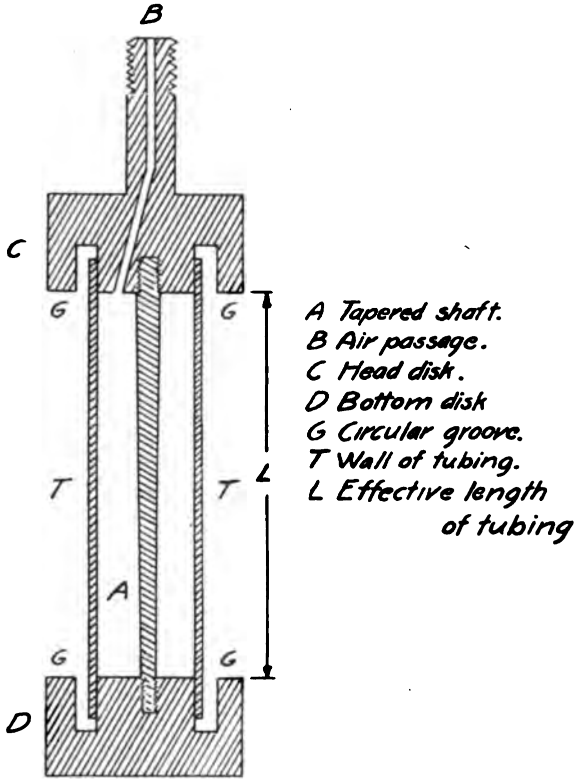

FIG. 7. CROSS-SECTION OF SPINDLE FOR MOUNTING A TUBE FOR COLLAPSE THE GROOVES *G* WERE FILLED WITH PACKING AND ASPHALTUM TO MAKE THEM WATERTIGHT

tube thus supported on the inside at each end was about one-eighth inch for the small tubes, and as large as one half-inch for the three-inch tubes. The groove left around the tube was for the packing. The end of the tube was wrapped with "friction" tape—the rubberized tape used for insulation in electrical wiring—and the surrounding groove was packed with it. Melted asphaltum was then poured over the tape, and in some cases the whole end was coated by dipping it into the asphaltum. There was seldom any trouble with leakage even at the highest pressures. The end constraint was, by this arrangement, reduced practically to that of the support of the internal plug. To eliminate the end pressure on the tube, an internal bolt was used which was threaded at both ends and screwed down by means of tapped holes at the centers of the end plugs. As this bolt did not come through the end plugs, a serious source of leakage was avoided. The bolt or shaft was tapered so that it could be easily withdrawn from the collapsed tube. The upper plug was turned down and threaded so as to screw into a hole tapped into the large steel-disk cover of the nickel-steel receptacle. A small canal through this connected the interior of the tested tube with a manometer. The pressure on the inside was atmospheric except at the moment of collapse. The steel and brass tubes were carefully machined to the required thicknesses, so that results have been obtained for several values of $\frac{t}{d}$ for each length of tube.

Three gauges were used, the connections being such that any one or all could be used in each case. Two of the gauges were made by Shaffer and Budenberg of Magdeburg, Germany, the third by the Crosby Gage Company. The first gauge read to a maximum of 300 kilograms per square centimeter, each scale division representing 5 kilograms per square centimeter; the second, to a maximum of 1000 kilograms per square centimeter, each scale division representing 10 kilograms per square centimeter; and the third to 2000 pounds per square inch, each scale division representing 50 pounds per square inch. These gauges were tested by using a Crosby Fluid Pressure Scale. This is the standard rotating piston balance made by the Crosby Gage Company for testing gauges up to a pressure of 25,000 pounds per square inch. The readings of the gauges were corrected in accordance with these tests; the corrections were so small that they could have been neglected for these experiments.

6. *Results.*—The results of the experiments are given in Tables 1 to 7 inclusive (pages 33 to 36) and in Figs. 8 to 14 inclusive of this bulletin. The number of lobes of the collapsed metal tubes is given in most cases. The number is recorded on account of its importance in the theory of stability and instability, which has been discussed by Southwell. In the collapse of glass tubes, we had the very striking

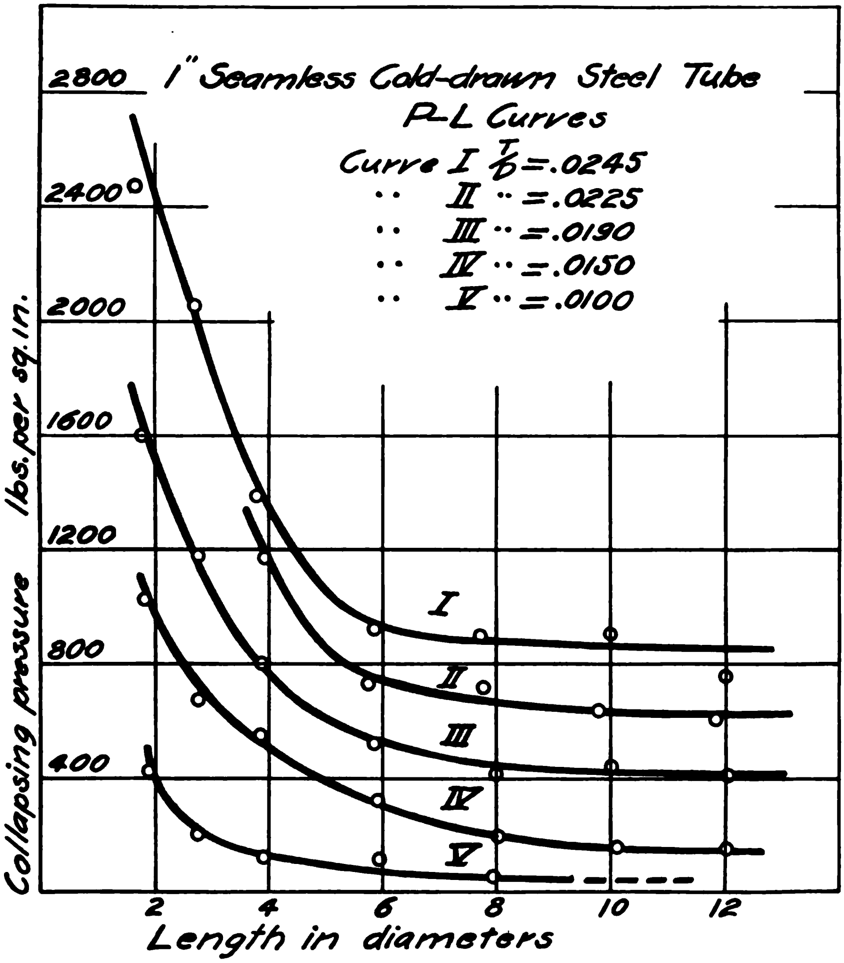

Fig. 8. Pressure-Length Curves for Five Different Thicknesses of One-Inch Seamless Steel Tubes

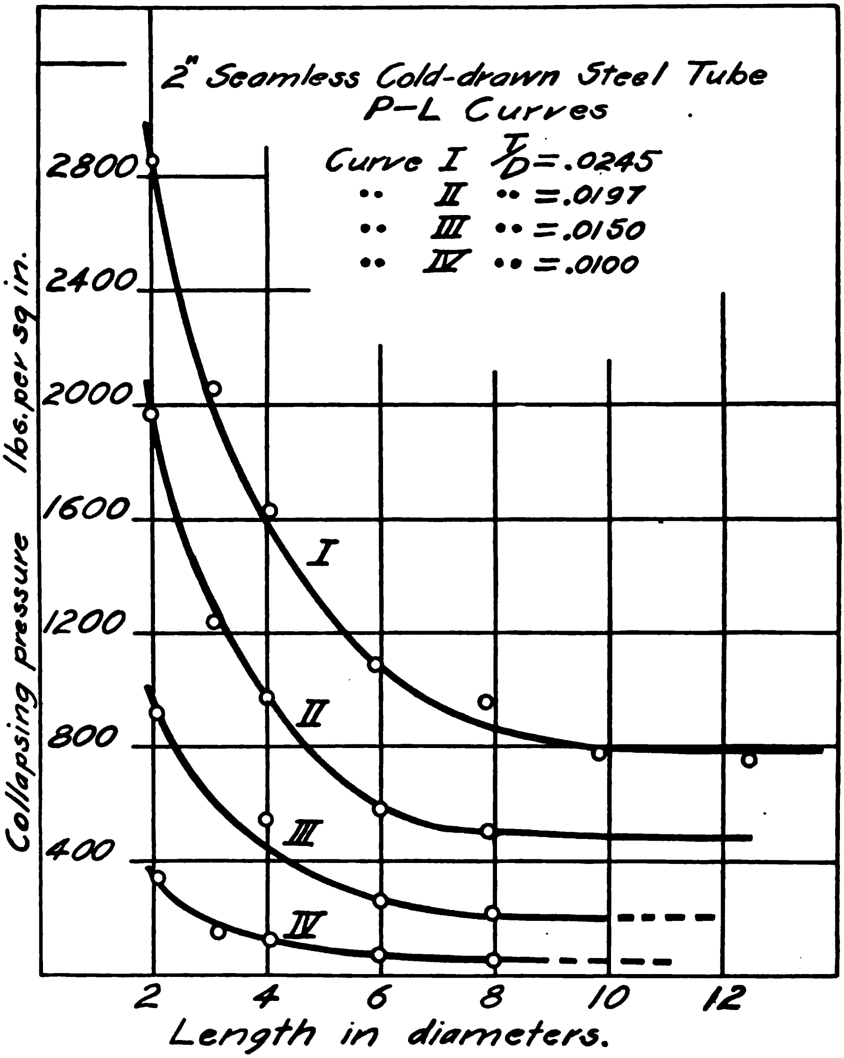

FIG. 9. PRESSURE-LENGTH CURVES FOR FOUR DIFFERENT THICKNESSES OF TWO-INCH SEAMLESS STEEL TUBES

occurrence of the glass being reduced to a fine powder. The data as given represent the results of the collapse of about one hundred and fifty tubes. In nearly all cases of the steel and the brass tubes, there were two tubes of the given dimensions; thus an average could be made for each point. This duplication also gave an immediate check on

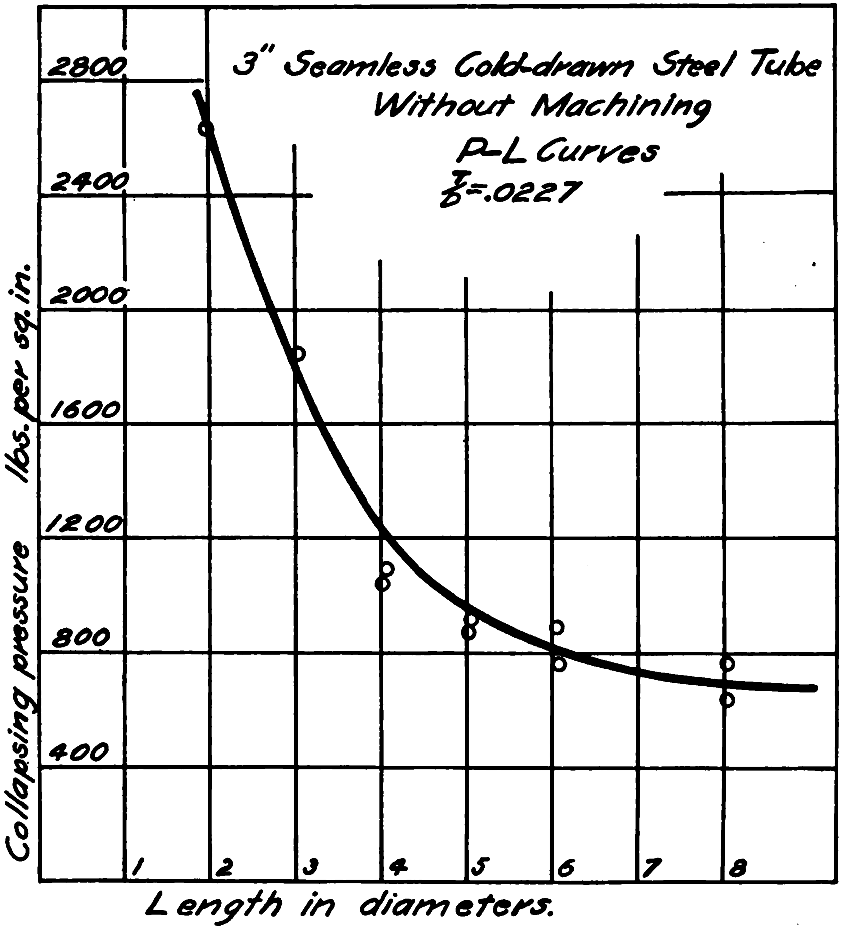

FIG. 10. PRESSURE-LENGTH CURVE FOR A THREE-INCH SEAMLESS STEEL TUBE WITH THICKNESS OF $\frac{t}{d}$ =.0227

freak collapses. Very few freak collapses occurred, however, and these could almost always be explained by the irregularities in dimensions or in material that appeared upon the inspection of the collapsed tube. While the machine work on these tubes was done with great care by the mechanicians of the department, Messrs. Hays and Buchanan, it was impossible to get the value of $\frac{t}{d}$ exactly the same each time. It was necessary therefore to make corrections in the observed collapsing pressures so as to have sets of results for a pressure-length curve with $\frac{t}{d}$ constant. These corrections were made by interpolation; it was assumed that the collapsing pressures varied as $(\frac{t}{d})^3$ for constant length. Since the total variations of $\frac{t}{d}$ in

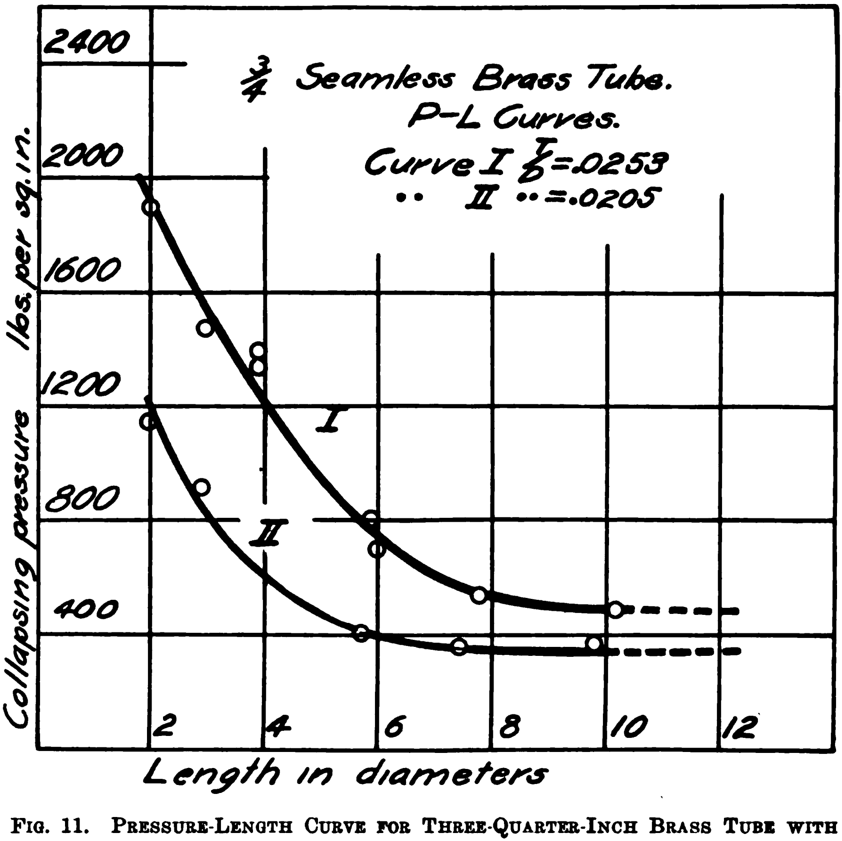

FIG. 11. PRESSURE-LENGTH CURVE FOR THREE-QUARTER-INCH BRASS TUBE WITH THICKNESS OF $\frac{t}{d}$ = .0205 AND .0253

these cases were very small, there is little assumption in the interpolations and corrections.

Observations were also made of about fifty steel tubes of half-inch and three-quarter-inch diameters. The results were very irregular, due partly to unavoidable variations in thickness, but more to a lack of homogeneity of the material. Although the tubes were called "seamless," they showed a longitudinal seam upon being machined, an evidence of their non-uniformity. An attempt was made to anneal some of the steel tubes to secure greater uniformity, but this introduced great irregularities of form. The tubes were packed inside an iron box and heated to about 1400 degrees F. (measured by a thermocouple) in the large gas furnace of the Mechanical Engineering Department and then allowed to cool slowly. The circular tubes came out quite elliptical, and annealing was therefore abandoned. The tubes used were purchased through dealers in the open market. The steel tubes were Shelby cold drawn steel tubes made by the National Tube Company. As these tubes are not always straight and are liable to vary in wall thickness, the dealers kindly helped us in selecting tubes of as perfect form as could be found in stock. Through the kindness

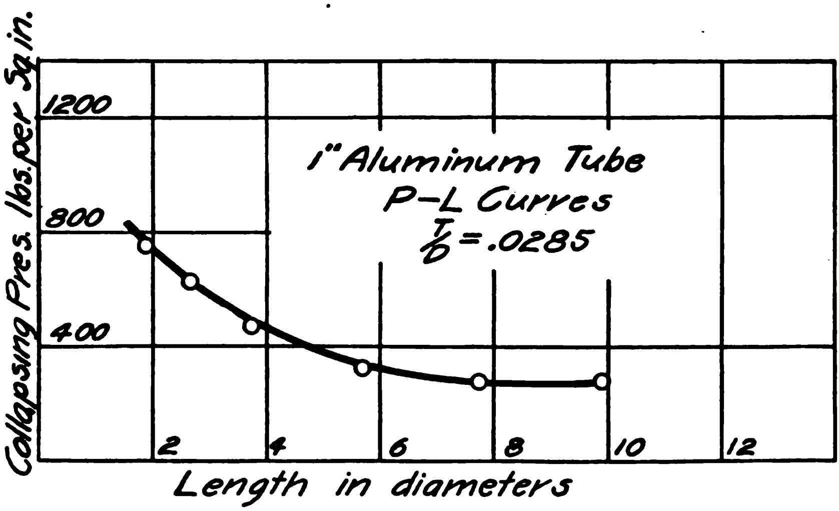

FIG. 12. PRESSURE-LENGTH CURVE FOR ONE-INCH ALUMINUM TUBE WITH THICKNESS OF $\frac{t}{d}$ = .0285

of the laboratory of Applied Mechanics of this University we got the following elastic constants of the steel and brass:

For steel,	Young's modulus . .	29.94×10^6 pounds per sq. in.
	Yield point . . .	$5.96^6 \times 10^4$ pounds per sq. in.
	Ultimate strength . .	7.25×10^4 pounds per sq. in.
For brass,	Young's modulus . .	13.90×10^6 pounds per sq. in.
	Yield point . . .	7.45×10^4 pounds per sq. in.
	Ultimate strength . .	9.90×10^4 pounds per sq. in.

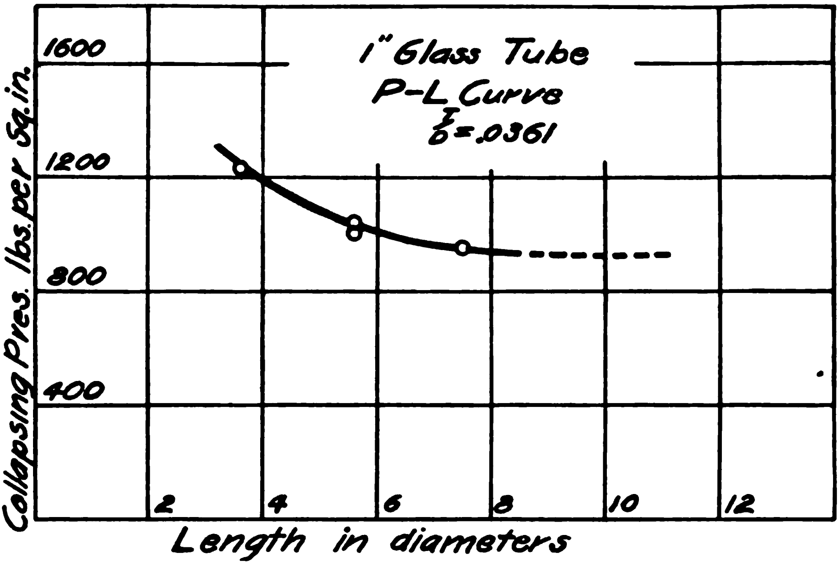

FIG. 13. PRESSURE-LENGTH CURVE FOR ONE-INCH GLASS TUBE WITH THICKNESS OF $\frac{t}{d} = .0361$

7. *Conclusions.*—A study of the curves leads to the following conclusions:

a. The pressure-length curves for different thicknesses and diameters are similar in shape. The series of curves for the one-inch and two-inch steel, the three-quarter-inch brass and the inch aluminum tubes show similar forms. The curves for glass and hard rubber differ from the others, the curve for the glass tubes bending more slowly, and that for the hard-rubber bending at a shorter length and more rapidly. The number of experiments on these last two materials is, however, too small to justify much generalization.

b. By taking, in the case of the metal tubes, the point for the length of six diameters, and drawing a hyperbola through this point using the equation $p' = p\frac{L}{l}$, there is in most cases a satisfactory agreement between the observed and the calculated curves for lengths of tubes less than six diameters. In Figs. 15, 16, and 17, the parts of the curves for lengths less than six diameters have been reproduced from Figs. 8, 9, and 10, and the calculated hyperbolas have been drawn on them. In a few cases the agreement of the two curves is very close; in one case the two curves are identical on the scale of the figure. In Figs. 18, 19, and 20, the observed curve and the calculated curves have been drawn and continued to lengths greater than six diameters. It is seen that the hyperbolas thus calculated show pressures less than those given by experiment. This length of six diameters, then, appears to be a "critical length." The experimental curve at this length bends rapidly towards the horizontal, particularly in the case of the thicker tubes. In the case of the thinnest steel tubes, both for those of one-inch and of two-inch diameters, that is, for tubes

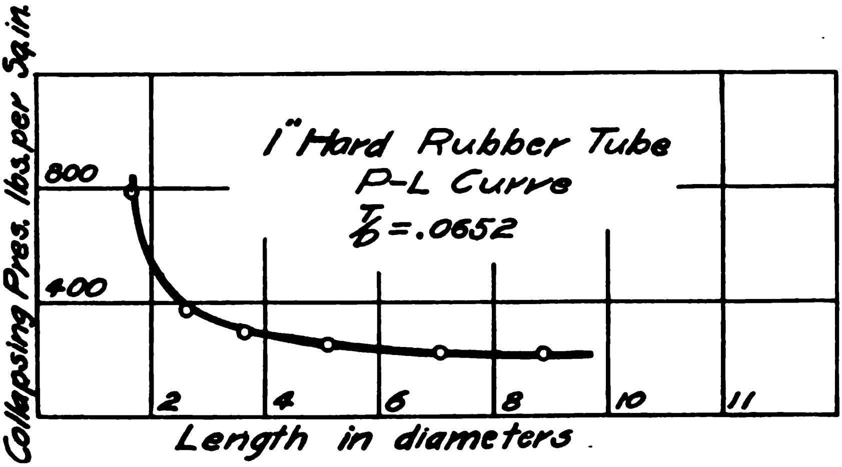

FIG. 14. PRESSURE-LENGTH CURVE FOR ONE-INCH HARD RUBBER TUBE WITH THICKNESS $\frac{t}{d} = .0652$

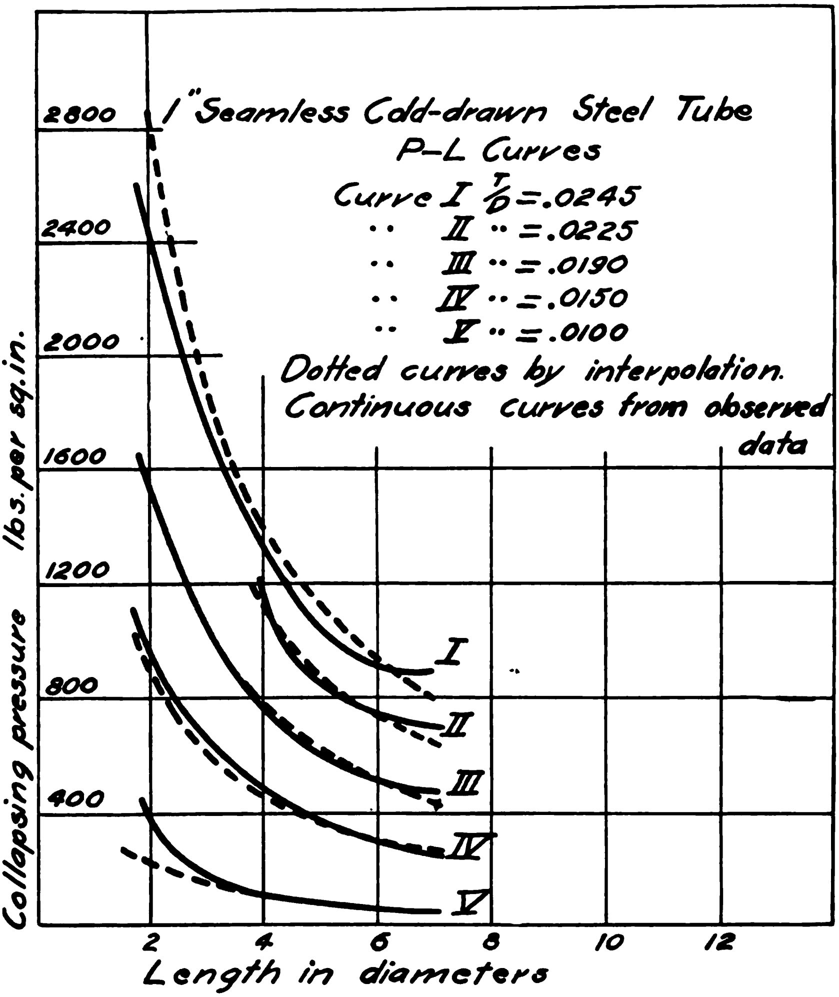

FIG. 15. PRESSURE-LENGTH CURVES OF ONE-INCH STEEL TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

in which the ratio of $\frac{t}{d}$ is about .001, the agreement with the hyperbola is not so exact; indeed, the maximum bend seems to occur at a much shorter length. While the curves for these very thin tubes show much uniformity, and the same characteristics are found in tubes of

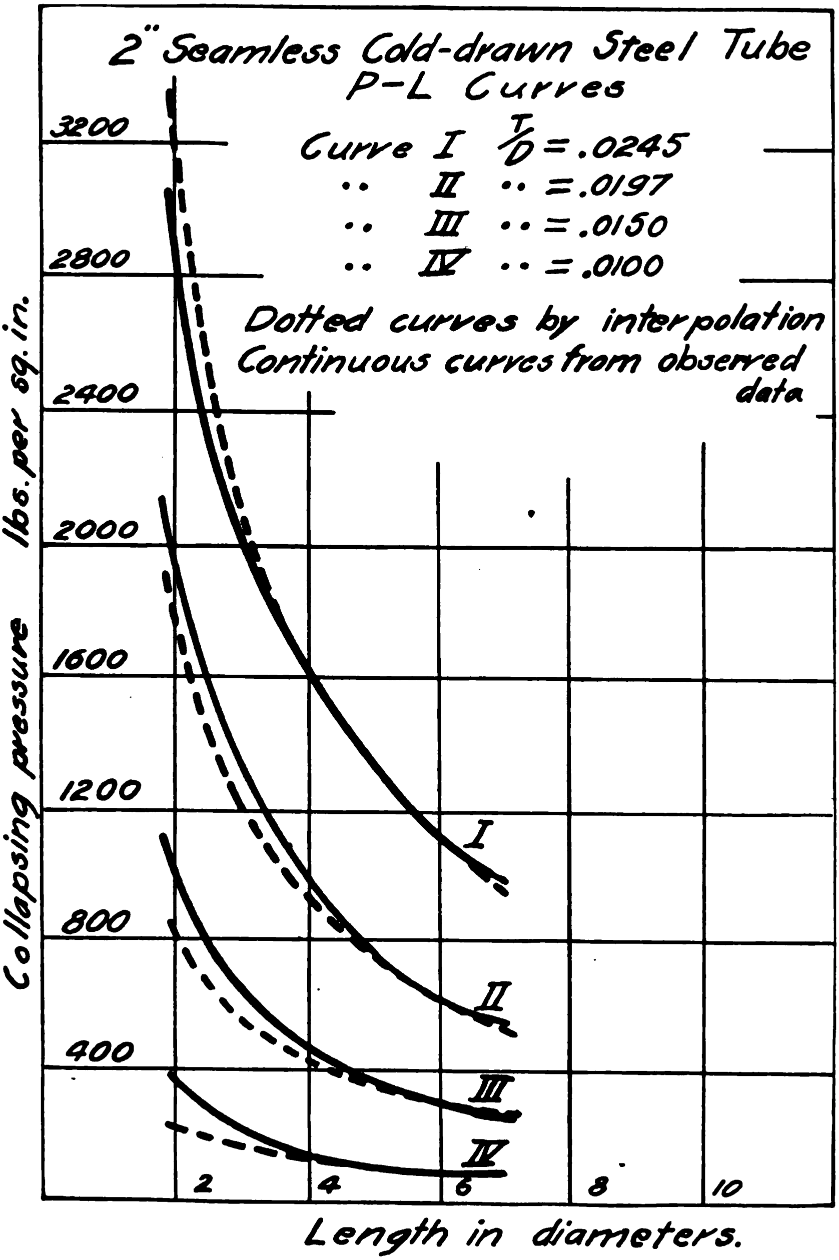

FIG. 16. PRESSURE-LENGTH CURVES OF TWO-INCH STEEL TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

different diameters, the percentage of error with such low pressures of collapse is necessarily greater.

c. The experimental curves of this investigation are not in agreement with the Southwell formula $L = k\sqrt{\frac{d^3}{t}}$ which Cook has used. This is shown in typical cases in Figs. 18, 19, and 20, where the curves for $L=1.75\sqrt{\frac{d^3}{t}}$ are drawn for the one-inch and three-inch steel tubes

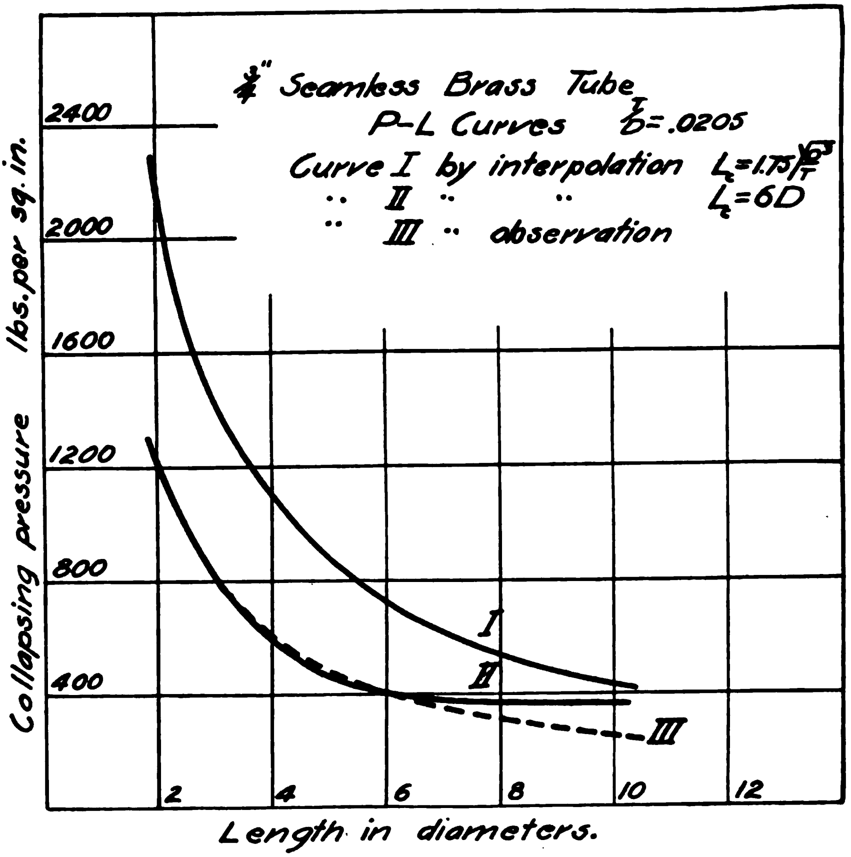

FIG. 17. PRESSURE-LENGTH CURVES OF THREE-QUARTER-INCH BRASS TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

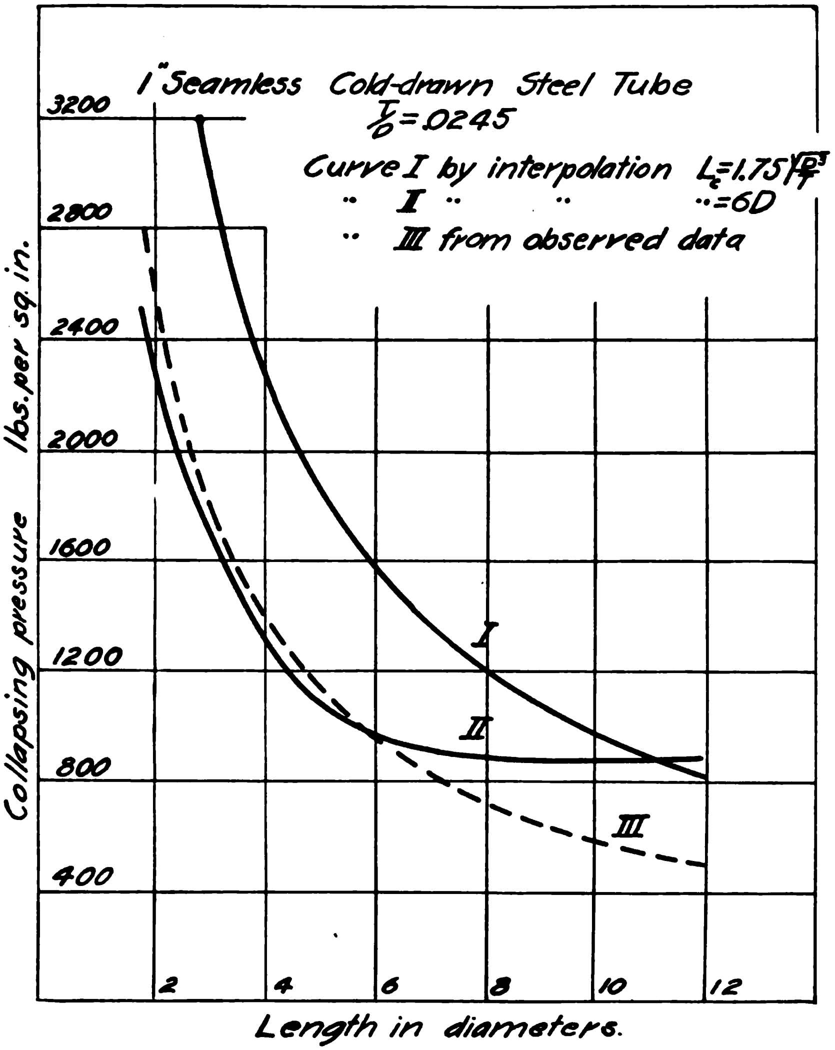

FIG. 18. PRESSURE-LENGTH CURVE OF ONE-INCH STEEL TUBE. EXPERIMENTAL CURVE COMPARED WITH TWO CALCULATED CURVES

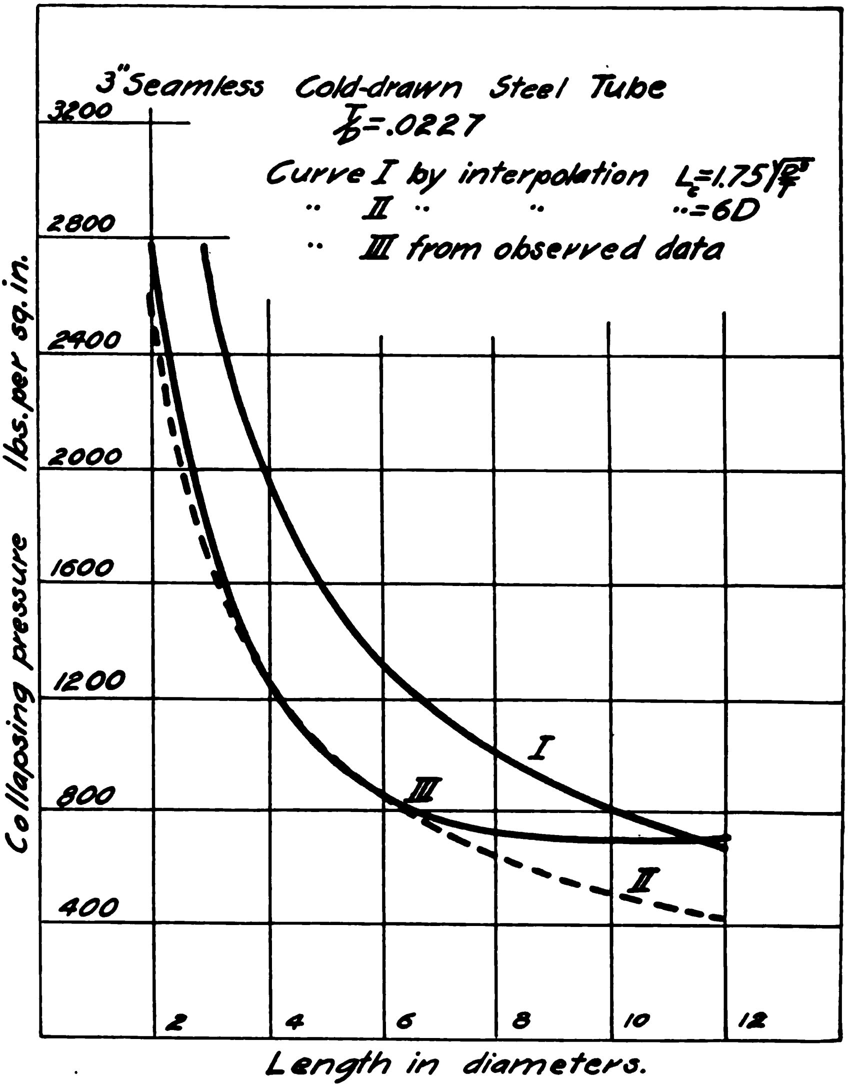

FIG. 19. PRESSURE-LENGTH CURVE OF THREE-INCH STEEL TUBE COMPARED WITH TWO CALCULATED CURVES

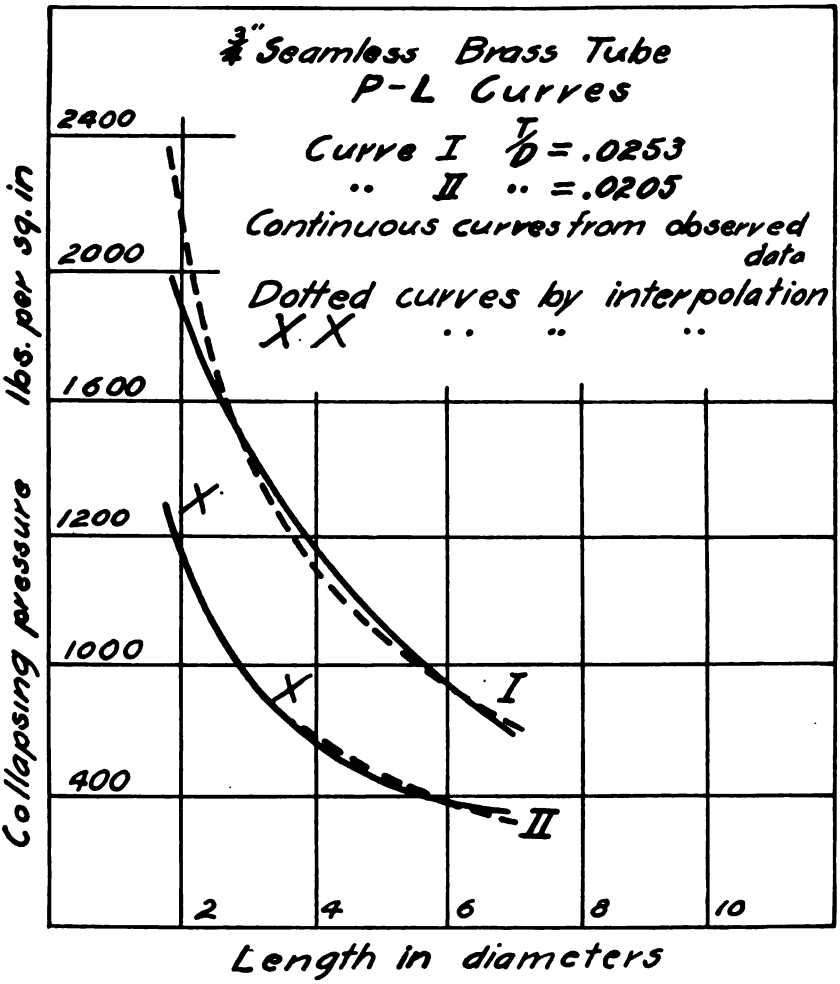

FIG. 20. PRESSURE-LENGTH CURVES OF THREE-QUARTER-INCH BRASS TUBES COMPARED WITH TWO CALCULATED CURVES

having a ratio $\frac{t}{d}$ of .0245 and .0227, and for the three-quarter-inch brass tube having a ratio $\frac{t}{d}$ of .0205. In the same figures the rectangular hyperbolas are drawn through the experimental point for a length of six diameters. If the formula $L = k\sqrt{\frac{d^3}{t}}$ is written in the form

$$L = \frac{kd}{\sqrt{\frac{t}{d}}} \quad . \; . \; . \; . \; . \; . \; . \; . \quad (11)$$

the critical length should increase directly as the diameter and inversely as the square root of $\frac{t}{d}$. It is evident that the length corresponding to the critical bend does vary as the diameter of the tube, but the curves do not show that the thinner tubes have longer critical lengths. Indeed our curves for the very thin one-inch and two-inch steel tubes rather indicate a shorter critical length than six diameters. To obtain reliable results on very thin tubes, facilities are required for testing tubes of large diameters, so that small variations in thickness and material may not affect the final results.

The experimental curves for these metal tubes for which the ratio $\frac{t}{d}$ is between .025 and .015 show that there is a "critical" bend at the length of about six diameters, the part of the curve for shorter lengths being approximately a rectangular hyperbola. Beyond this critical bend the curve approaches more or less rapidly a straight line parallel to the axis of lengths. These results, in connection with previous results on the collapsing pressures of long tubes, make it possible to calculate from the data with reasonable approximation the collapsing pressures of short tubes of certain thicknesses.

TABLE 1

ONE-INCH STEEL TUBES

INSIDE DIAMETER .942 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.72	1.8	.0245	2490	4
1.74	1.8	.0190	1600	3 or 4
1.785	1.86	.0150	1030	3 or 4
1.78	1.87	.01	420	4
2.63	2.73	.0245	2060	3
2.63	2.74	.0190	1180	3
2.63	2.74	.0150	670	8
2.63	2.75	.010	200	3
3.69	3.82	.0245	1390	2 or 3
3.69	3.84	.0190	800	3
3.69	3.85	.0150	550	3
3.69	3.9	.010	120	3
3.72	3.9	.0225	1170	3
5.63	5.85	.0245	920	2
5.63	5.73	.0225	730	2
5.63	5.85	.0190	520	2
5.63	5.9	.0150	320	2
5.63	5.9	.010	115	2
5.5	5.6	.0227	675	3
7.5	7.8	.0245	900	2
7.59	7.7	.0225	710	2
7.56	8.	.010	50	2
7.44	7.8	.015	210	2
7.62	8.	.015	190	2
7.62	8.	.019	420	2
9.6	10.	.0245	900	2
9.56	9.75	.0225	625	2
9.6	10.	.019	440	2
9.56	10.	.015	150	2
11.6	12.	.0245	750	2
11.63	11.85	.0225	600	2
11.63	12.	.019	375	2
11.6	12.1	.015	145	2

TABLE 2

TWO-INCH STEEL TUBES

INSIDE DIAMETER 1.873 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
3.91	2.	.0245	2870	3
3.88	2.	.0197	1770	3
4.	2.1	.015	920	3
3.88	2.1	.010	350	2
5.94	3.1	.0245	2100	3
5.94	3.1	.0197	1250	3
5.94	3.1	.015	580	3
5.94	3.1	.010	150	2
7.8	4.1	.0245	1630	2
7.63	4.	.0197	980	2
7.63	4.	.015	540	2
7.66	4.	.010	135	2
11.34	5.9	.0245	1080	2
11.34	6.	.0197	590	2
11.34	6.	.015	260	2
11.34	6.	.010	70	2
15.13	7.9	.0245	960	2
15.13	7.9	.0197	510	2
15.13	8.	.0150	220	2
15.13	8.	.0099	55	2
18.94	9.9	.0245	794	2
23.0	12.5	.0245	780	2
27.8	14.5	.0245	855	2

TABLE 3

THREE-INCH STEEL TUBES

INSIDE DIAMETER 2.87 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
5.88	2.	.0227	2640	4
9.12	3.1	.0227	1960	..
12.14	4.1	.0227	1390	2
12.17	4.1	.0227	1040	2
12.19	4.1	.0227	1010	2
15.06	5.1	.0227	873	2
15.06	5.1	.0227	910	2
18.13	6.2	.0227	756	2
18.28	6.2	.0227	895	2
24.13	8.2	.0227	684	2
24.25	8.2	.0227	770	2

TABLE 4

THREE-QUARTER-INCH BRASS TUBES

INSIDE DIAMETER .751 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.56	2.	.0138	310	3
1.56	2.	.0138	400	3
1.56	2.	.0205	1150	3
1.56	2.	.0253	1900	3
2.28	3.	.0138	345	..
2.28	3.	.0138	300	..
2.28	2.9	.0205	920	..
2.25	2.9	.0253	1470	..
2.25	2.9	.0253	1310	..
3.	3.9	.0253	1400	..
3.	3.9	.0253	1360	..
3.	3.9	.0276	1420	..
4.	5.7	.0205	409	2
4.	5.7	.0205	429	2
4.56	5.9	.0253	810	2
4.63	6.	.0253	700	2
5.69	7.4	.0205	348	2
6.	7.8	.0205	550	2
7.5	9.8	.0205	362	2
7.5	9.8	.0205	388	2

TABLE 5

ONE-INCH ALUMINUM TUBES

INSIDE DIAMETER .944 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.18	1.9	.028	750	4
2.59	2.7	.0286	630	3
3.63	3.7	.0285	470	3
3.63	3.7	.029	475	3
5.53	5.7	.0288	330	2
7.53	7.8	.029	280	2
9.50	9.9	.0295	280	1

TABLE 6

ONE-INCH GLASS TUBES

INTERNAL DIAMETER .96 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Condition of Collapsed Tube
3.59	3.7	.036	1230	practically pulverized
5.63	5.8	.036	1040	" "
5.63	5.8	.036	1000	" "
7.47	7.7	.036	950	" "

TABLE 7

ONE-INCH HARD RUBBER TUBES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Condition of Collapsed Tube
1.69	1.6	.0643	790	One side caved in and broken in very fine parts.
2.63	2.5	.0654	370	In irregular parts
3.63	3.4	.0650	290	"
5.56	5.2	.0654	250	"
7.56	7.1	.0652	220	"
9.55	8.9	.0645	220	"

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PERCENTAGE OF EXTRACTION OF BITUMINOUS COAL WITH SPECIAL REFERENCE TO ILLINOIS CONDITIONS

BY

C. M. YOUNG

ILLINOIS COAL MINING INVESTIGATIONS
COÖPERATIVE AGREEMENT

(This Report was prepared under a Coöperative Agreement between the Engineering Experiment Station of the University of Illinois, the Illinois State Geological Survey, and the U. S. Bureau of Mines)

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UNIVERSITY OF ILLINOIS
ENGINEERING EXPERIMENT STATION

BULLETIN No. 100 JUNE, 1917

PERCENTAGE OF EXTRACTION OF BITUMINOUS COAL WITH SPECIAL REFERENCE TO ILLINOIS CONDITIONS

BY
C. M. YOUNG
ASSISTANT PROFESSOR OF MINING RESEARCH

ENGINEERING EXPERIMENT STATION
PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

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PERCENTAGE OF EXTRACTION OF BITUMINOUS COAL WITH SPECIAL REFERENCE TO ILLINOIS CONDITIONS

INTRODUCTION

1. *Preliminary Statement.*—The purpose of the discussion presented in this bulletin is to record the results now being obtained in recovering coal in the mines in Illinois and in other bituminous coal mining districts of the United States. A brief discussion is also presented with reference to recovery in the principal European countries. Where the methods employed are now producing an unusually good percentage of extraction, the conditions under which the mining is carried on are described in considerable detail with the belief that they may suggest changes in practice which will be helpful to those who are now endeavoring to recover a greater percentage of the coal in the ground.

Most of the data presented were obtained from those operating the mines, and represent, therefore, calculations or estimates based upon thorough familiarity with conditions. Some of the methods by which high extraction is attained in other districts are described with the hope that the coal producers of Illinois may find herein suggestions which will prove helpful in their efforts to attain higher recovery.

It has been impossible to include in a single publication all the material available concerning the physical conditions encountered and the methods adopted in the various coal fields, but there will be found in the bibliography a list of books and articles in which these subjects are covered in greater detail. It is the present purpose to begin at an early date a more extended investigation of the plans and dimensions of mine workings in Illinois with reference to the cost of production and the percentage of extraction.

2. *Acknowledgments.*—The writer wishes to acknowledge his indebtedness to **Professor** H. H. **Stoek**, head of the Department of Mining Engineering, University of Illinois, and to **Mr.** F. W. **DeWolf**, Director of the Illinois State Geological Survey, and also to **Mr.** G. S. **Rice**, Chief Mining Engineer, United States Bureau of Mines, under whose direction the work of the Illinois Coal Mining

Investigations is being carried on. Professor Stoek has been especially helpful in the collection of material for the present bulletin. Many operators and engineers throughout the country have contributed statements concerning the districts with which they are familiar. The state mine inspectors have assisted in the work by suggesting mines at which particularly good records of extraction have been made and also mines at which new methods are being tried with a view of increasing the percentage.

3. *Summary.*—The facts and information presented in this bulletin include:

(1) A general statement of the importance of the problem of increasing the percentage of extraction of the coal in the ground in order to utilize the coal resources to a greater extent than at present, and, if possible, to decrease the cost of producing coal; also an account of previous efforts made to compile data upon this subject.

(2) A statement with reference to the conditions which have influenced the development of American coal mining methods and which must be considered in changing these methods in order to obtain more nearly complete recovery.

(3) A record of the recovery of coal in Illinois in the past, and a discussion of the efforts now being made to increase the percentage of extraction.

(4) An account of methods adopted in other states and in certain European countries by which higher percentages of extraction are being obtained.

(5) A brief history of the development of English mining practice, upon which American practice is founded.

(6) A short bibliography with reference to the subject of coal mining methods.

4. *Conclusions.*—A summary of conclusions suggested by a study of the data and information contained herein is presented as follows:

(1) In general in America probably one ton of coal has been left in the mine for every ton brought to the surface.

(2) An effort is being made in many sections of the United States and in a number of Illinois mines to decrease this loss of coal.

(3) The low percentage of recovery in the United States is largely due to economic conditions and to efforts to produce cheap coal.

(4) Where economic conditions have been favorable, percentages of recovery have been obtained in the United States quite as high as in any of the foreign countries in which usually the economic conditions have not been such as to make the production of cheap coal the determining element in the choice of a method.

(5) The low price at which much of the coal land in the United States has been bought has not offered an inducement to save the coal.

(6) The best results in recovery are now being obtained in districts where the value of coal land is high.

(7) As a general rule better extraction is being obtained in West Virginia and Pennsylvania than in the Middle West.

(8) In view of the results being obtained in some other districts, under conditions no more favorable than those in Illinois, the percentage of extraction in Illinois should be increased.

(9) The best results are being obtained by the larger and stronger companies which can afford to plan for the future.

(10) The low value of the smaller sizes of coal in the past has been a drawback to pillar drawing, because very often pillar coal has contained more of the small sizes than room coal. With the increasing use of small sizes in mechanical stokers, the price will undoubtedly advance to nearly the same level as that of the larger sizes; thus this drawback to greater recovery will gradually disappear.

(11) One of the reasons why newer methods have not been generally tried is to be found in the prejudice, too common in coal mining practice, against innovations, and in the fact that mining methods have been based largely upon previous practice in other countries or in other states.

(12) Many of the attempts to draw pillars have been unsystematic. Upon such unsystematic work are based many of the opinions concerning the technical and commercial practicability of pillar drawing and the prejudices against it.

(13) Subsidence of the surface must be regarded as a necessary accompaniment of mining. Instead of trying to prevent

subsidence, the pillars should be removed systematically so that the surface subsidence will occur uniformly and not in isolated spots. Although there may be a temporary disturbance of the surface, after a short time its condition will be as good or nearly as good as before the mining.

(14) In Illinois, at the present time, more than fifty per cent of the coal is frequently left in the ground in an effort to prevent squeezes and subsidence; even then it is not at all certain that the desired result is accomplished.

(15) The best results may be obtained by driving room entries to their full length, then by beginning the rooms at the inby end of the entry, in order that pillar drawing may begin as soon as the inby room is finished.

(16) To be effective, pillar drawing must begin as promptly as possible after the rooms are worked out.

(17) Where pillars are left to be drawn subsequently, the coal is usually lost, because the pillars are crushed through squeezes, or because it is not found economical or convenient to take the coal out and at the same time to keep up the output of the mine with the compartively small amount of coal left. In other words, unless pillar drawing follows very closely after the first working, very little pillar coal is obtainable.

(18) In many districts poor top has prevented taking out the full thickness of the coal, and one of the great losses is that due to coal left in the roof. This loss has been overcome in some cases very successfully, and should be carefully studied.

(19) The reported percentages of extraction are usually too high because, in estimating, often only the section mined is considered and no account is taken of top or bottom coal left unmined. Also frequently only limited areas of the mine are considered instead of the mine as a whole.

(20) At different mines in the same region where physical conditions are practically the same, the mining methods vary widely with regard to length of rooms, number of rooms in a panel, thickness of barrier pillars, etc. This variation in practice suggests the advisability of a detailed study to determine, if possible, a standard method for a given set of conditions.

CHAPTER I

MINING METHODS AND CONDITIONS IN RELATION TO EXTRACTION

5. *Introduction.*—The subject of the percentage of coal extracted from the mines in the United States has received very meager attention, except in the case of individual mines or companies. The only comprehensive official study of an extended coal mining area has been in the anthracite district of Pennsylvania, where the high value of the coal and the knowledge that the supply is limited early stimulated an interest in the subject. This interest led to the appointment of the Coal Waste Commission which reported in 1893.*

In 1905 H. H. Stoek† published a table of coal pillar data which contained percentages of extraction gathered largely by correspondence. See Table 1, page 23.

In 1914 A. W. Hesse‡ collected as much information as possible on this subject, which is summarized in Table 2, page 24.

In previous bulletins of the Coöperative Coal Mining Investigations tables of pillar data and percentages of extraction were given. These are summarized in Table 3, pages 25, 26, 27.

Doubtless many of the figures in these tables and others on the percentage of recovery are open to question, but they represented the best and most nearly complete information available when they were published. There are several reasons for questioning the accuracy of the figures on extraction. Chief among them is the fact that estimates are usually based upon areas which are too small or upon insufficient data. A single panel or a single lease is sometimes used as a unit upon which to base estimates, and often the areas thus selected are favorably located. While the estimates may represent results obtained with the given method of mining, they by no means represent the average results for the mine as a whole. The panel selected for measurement is usually one in which there has been no squeeze, while all about it there may be squeezed areas in which large amounts of coal have been lost. Many estimates are based upon possible future recovery from pillars, which may or may not be obtained.

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*Report of Commission Appointed to Investigate the Waste of Coal Mining with the View to the Utilizing of the Waste, 1893.

†Mines and Minerals, Vol. 26, p. 107, 1905, and International Library of Technology, Vol. 150, par. 40, p. 60.

‡"Maximum Coal Recovery," W. Va. Min. Inst., June 3, 1914; Coll. Eng., Vol. 35, p. 13; and Coal Age, Vol. 5, p. 1051.

Estimates covering extraction frequently do not take account of top and bottom coal left in the mine, and the values reported often refer only to the section of the coal actually mined. In mining a coal bed ten feet thick, for instance, two feet of top coal may be left unmined. The maximum percentage of extraction from mining eight feet, in this case, would be eighty per cent of the total coal in the bed. If, then, fifty per cent of the eight feet mined is obtained, only forty per cent of the total coal in the bed is recovered.

The only accurate method of estimation is to divide the actual amount of coal mined, as determined by the tonnage for which the miner is paid, by the amount of coal in the ground as determined by multiplying a given area by the average thickness shown in a large number of sections of the bed.

Even where great care is exercised, results are often subject to errors. The causes for these have been outlined by Smyth* as follows: Inaccuracies in railroad weights of possibly five to ten per cent, inaccuracies in estimation of coal used at the mines frequently amounting to ten per cent, inaccuracies in estimating the mean thickness of the bed amounting, even in very uniform beds, probably to five per cent, difficulties of obtaining final figures until a mine is worked out.

The present condition of the coal mining industry in this country is a natural result of the course and character of its development. In general, only those beds and even parts of beds have been worked, the exploitation of which would result in the largest immediate profits. Those methods of mining which were cheapest and which promised the largest profit on the coal produced have been followed, often without regard to the possible injury of the mine or the resulting loss of coal. There has been, moreover, no restriction of market, and in many cases districts have been opened when there has been very little demand for coal in the surrounding territory, but when conditions of operation and transportation have been such as to make it possible for coal from these districts to enter markets already supplied. The result has been cheap coal, produced by wasteful methods.

Another result has been over-development of the industry. The opening in nearly all districts of too many mines has resulted in the idleness of many mines during a large part of each year with the accompanying increase in the cost of production. In dull periods coal has frequently been sold for less than the cost of production in order that mines might be kept in operation and certain fixed charges met. This subject was taken up by Bush and Moorshead in 1911 in a paper* before the American Mining Congress in which it was said that the production in this country exceeded the consumption first in 1891, and that the difference between consumption and capacity for production had steadily increased. The strike of 1910 in Illinois, Indiana, and the Southwest emphasized the over-capacity of the mines of that region. Though the mines of Illinois were idle during six months of the year, the production of 45,900,246 tons was only ten per cent less than the production of the previous year. The mines of Oklahoma, Arkansas, and Missouri were also idle during six months of 1910 because of the strike, but the production showed an average decrease of only twenty per cent. It was also said that the possible capacity of West Virginia mines was seventy-five per cent more than the total production, that the output in the Pittsburgh and the No. 8 Ohio districts was reduced to thirty per cent of the normal production during the three or four months of each year when navigation on the lakes was closed, and that few properties during the three preceding years—1909, 1910, and 1911—had been operated more than 225 working days per year.

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* Smyth, John G., Personal Communication.

This over-production, with its small profit or even loss in the operation of mines, results in a natural tendency to employ only those methods which will insure cheap coal. It is natural, also, that under these conditions there should exist an attitude of hesitancy with regard to the adoption of new or different methods. Neither the coal producer nor the public has as yet become aroused to the full realization of the fact that the natural resources of the country are not inexhaustible. The coal mining engineer of America accordingly, has not had as his problem the development of methods of extraction which would result in the largest percentage of ultimate recovery, but rather the development of methods which would result in the lowest cost of production. In many cases, however, as is shown by the detailed descriptions given later, where economic conditions have seemed to warrant it, methods have been developed by

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* Bush, B. F., and Moorshead, A. J., "The Condition of the Bituminous Coal Industry," Proc. Amer. Min. Cong., p. 246, 1911.

American engineers and coal producers which have given a percentage of recovery equal to that secured in any European country.

The fact should be borne in mind, when comparisons are made between mining methods in different countries, that, while it is true that the percentage of extraction is less in this country than in most of the European countries, the cost of coal to the consumer and the profit to the producer are also less.

The subject of the comparative cost of production of coal and of the comparative profits realized in Great Britain and in the United States was taken up by Rice* substantially as follows: the average value of coal in the United States on cars at the mine in 1913 is reported as $1.18 per short ton for bituminous coal and $2.13 per short ton for anthracite. In Wales, in 1913, the average value per short ton at the mines for all kinds of coal was $2.55, and in Great Britain as a whole, $2.21. In the German Empire the average value for all kinds was $2.27, and for Westphalia it was $2.37 per short ton. Net mining profits in Great Britain and in Germany are between twenty-five and fifty cents per ton, while profits in the United States for bituminous coal are probably not more than five cents per ton.

It is a matter of course that more expensive methods of mining cannot be adopted without increasing the cost of the coal, and under the conditions which have prevailed in the coal industry for many years there could be no material increase in the cost of coal to the producer without a corresponding increase in the selling price. The prevailing opinion, however, that the percentage of recovery cannot be greatly increased without an increase in the cost of production is questionable, and certainly this increase in cost would not be as great as is generally believed. This is a matter which can be conclusively determined only by actual trial of new methods extending over a sufficient period to insure the reliability of the results. The fact that the adoption of methods which result in an increase in the percentage of extraction has been possible in some districts with little or no increase in cost at least furnishes a reason for thinking that similar changes could be made in other districts with similar results.

Careful planning of operations over long periods and steady working are necessary in order to obtain a high percentage of extraction. At present these conditions are impossible in many districts and can be attained only by centralized control of production and selling price, which will provide against alternation of idle and rush periods with the disorganization which accompanies them. Under existing conditions it is feared by operators that the necessary coöperation would be interpreted and attacked as a violation of anti-trust laws. In some of the European countries syndicates working in coöperation with the governments regulate the output of the mines and the selling price of coal with results which are said to be highly satisfactory and conducive to a high recovery.*

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*Rice, G. S., "Mining Costs and Selling Prices of Coal in the United States and Europe, with Special Reference to Export Trade," Second Pan-American Scientific Congress.

One of the chief commercial factors affecting the choice of a method has been the cost of coal in the ground. This has generally been very low, and the loss of coal, therefore, has not been considered a serious matter. Even at the present time the value of coal rights in the southern Illinois field, where the No. 6 bed is worked, is estimated at not more than $100 to $150 an acre, and it has been only a very short time since such coal rights could be purchased for less than $50 an acre. The thickness of this coal is somewhat variable, being in some places fourteen feet or more, but, if we assume that only about seven feet is worked, the output will amount to about 12,000 tons per acre and the cost of coal in the ground will be about one cent per ton. A great deal of the coal in the state, however, has been bought at a very much lower figure. In some cases also there is a second bed of coal which will be available later, and when this is considered, the cost of coal in the ground will be much less than one cent per ton.

This phase of the subject was discussed by Rice,† in 1909, as follows:

"The influencing conditions causing the great losses that are at present incurred are:

1. Cheapness of 'coal in place'; that is, in the seam.
2. Low market prices, resulting from extreme competition.
3. Character of the seam, roof, and floor as determining the method of mining.
4. Surface subsidence due to mining.
5. Interlaced boundary ownerships.

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* Scholz, Carl, "The Economics of the Coal Industry," Proc. Amer. Min. Cong., p. 241, 1911.

†Rice, George S., "Mining-Wastes and Mining-Costs in Illinois," Trans. Amer. Inst. Min. Eng., Vol. 40, p. 31, 1909.

6. Carelessness in mining operations.

The first two factors, taken together, are the controlling ones in most mining operations in influencing the choice of a mining system. The majority of Illinois operators are sufficiently progressive to find ways and means to take out practically all the coal under a given area if it could be made evident that it paid to do so. That many do not do all that can be done in this direction is apparent; but if, without unusual investment, a profit of operation could be shown in taking out all the coal over the profit made by present methods, the industry could undoubtedly find men to accomplish the task. In other words, from an engineering standpoint practically all the coal under a given area can be taken out. It is a question of cost.

"*Cheapness of Coal in Place.*—This is chiefly due to the great abundance of coal. Except in the barren northern one-fourth of the State, lying north of the outcrop of the coal-basin, the development of a tract depends primarily not on the possibility of finding coal in that particular locality, but on the question whether it is a suitable place, from a market standpoint, to open a mine, the thickness of seam and the quality of the coal being considered.

"The price of coal rights varies from $10 per superficial acre in the middle part of Illinois, away from the mining centers to $100 per acre near developed mines. Or, in the case of leasing, from 2 cents per ton run-of-mine hoisted, in the southern part of the State, to 5 cents in the northern part. The cost of the fee is relatively so much cheaper per ton than leasing that the latter system is not much used. The ownership of the coal by the operator is conducive to better mining, but relative to other items that go to make up the total, the cost of the 'coal in place' is so low as to be almost negligible. In central Illinois, in some cases, at a cost of only $10 per acre, two workable seams, from 6 to 8 ft. thick, are obtained. Allowing only 50 per cent yield of the two seams, 13,000 tons would be produced per acre, the purchase cost thus being 1/13 of a cent per ton, or about 1/1000 of the total cost of production in central Illinois. In the Wilmington long-wall field the average cost of the coal rights is about $50 per acre. The seam there, although it averages a trifle less than 3 ft. in thickness, produces about 5,000 tons per acre. The cost is therefore about 1 cent per ton in place, which is 1/130 of the total cost of production. Hence, it may be seen that there is little incentive, from the standpoint of the purchase price of the coal, to save the latter in mining operations."

The cost of coal rights has very greatly increased since Rice's discussion, and there is every reason to believe that the value of coal in the ground will be much greater in the future than it has been in the past. During most of the productive period, however, the coal in the ground over a considerable part of the State has been worth not more than one-tenth of a cent per ton, and under these circumstances the loss of coal has, naturally, not been considered a serious matter. What has been important, and still is, is the extraction of coal at low cost, and the subject of high recovery is one of increasing importance at the present time.

Every ton of coal left in the ground represents the loss of a possible profit. Every ton of such coal represents a loss in increased value. An acre of coal left in the ground at any time means the extraction of another acre at some later time when the value of coal in the ground will be greater. In other words, producers are now extracting coal, worth possibly $150 an acre, which might be left until it would reach even a greater value if it were not for the fact that coal was wasted in the ground when it was worth only fifty dollars an acre.

A low percentage of extraction increases the cost of production, because, for a given output, the workings must cover a larger area. This involves longer haulage roads, and, consequently, a greater investment in rails and trolley wire, greater maintenance expense, greater consumption of power, lower output per unit of equipment, and lower output per man. With long haulage roads there is greater chance for derangements of the track or for falls of roof, which may cause the stopping of haulage until the trouble is removed or may result in wrecks if the trouble is not discovered in time. Another source of danger lies in the greater haulage speed which must be employed on long roads if the output is to be maintained.

The cost of ventilation is also higher, because these larger workings require a larger quantity of air to maintain safe conditions, and more power is required to circulate air through the longer passages. There is, moreover, a greater loss of power because of the more numerous stoppings, which are often inefficient.

Another difficulty accompanying the spreading of the workings over a large area is that of providing the intensive supervision which is highly desirable in coal mining, particularly where skilled workmen have been replaced by comparatively unskilled laborers, and the pick has been replaced by explosives and mining machines. Unless the cost of operation is to be increased by the employment of a larger number of foremen or face bosses, this intensive supervision can be obtained only by concentration of the workings.

6. *Subsidence.*—One reason for the use of methods involving low extraction is the desire to maintain the original surface of the ground. Rice* discussed surface subsidence as follows:

> "The influence of this factor upon the yield results from the high value of Illinois lands for agricultural purposes. . . . If the long-wall system were applied to the thick seams, when applicable at all, it would cause a considerable derangement of the surface, and when the latter is so nearly level as the prairie-land of Central Illinois, it makes the question of subsidence a serious one. . . . However, until the agricultural land in the United States becomes insufficient to fill the needs of the population, which would be reflected in a continual increase of price for farming land, the money-loss from temporarily destroying the surface in places is relatively small, as compared with the selling price of the coal mined from the seam. Taking the average value of the surface at $125 per acre, if 80 per cent be rendered worthless the immediate money-loss would be $100 per acre. A seam 6 ft. thick would contain per acre 11,000 tons of coal in place, yielding, at 90 per cent, 9,900 tons. The damage done by practically destroying the surface would be only 1 cent per ton. If the land-prices should rise to an amount two or three times as great as the value stated, this loss would still not prohibit mining."

As far as the long-wall district is concerned, very little if any damage has resulted from subsidence, and little attention has been given to the subject. The most noticeable effects are generally temporary, and farm operations are not hindered.

The subject of the relation of surface values to subsidence in Illinois has been considered by L. E. Young in Bulletin 17 of this series.† Fig. 1 is a map reproduced from this bulletin, on which the approximate values of farm lands are shown. Table 4, page 28, shows the relation between coal values and surface values. The land values indicated on the map and set forth in the table suggest that it might be possible, at least in many cases, to mine coal at a small profit even if the value of the surface were totally destroyed. There is no need, however, for assuming the permanent destruction of the surface or even its serious permanent injury. Generally any damage resulting from subsidence could be largely or wholly re-

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*Rice, Op. cit., p. 40.

†Young, Lewis E., "Surface Subsidence in Illinois," Ill. Coal Min. Invest., Bulletin 17, p. 52, 1916.

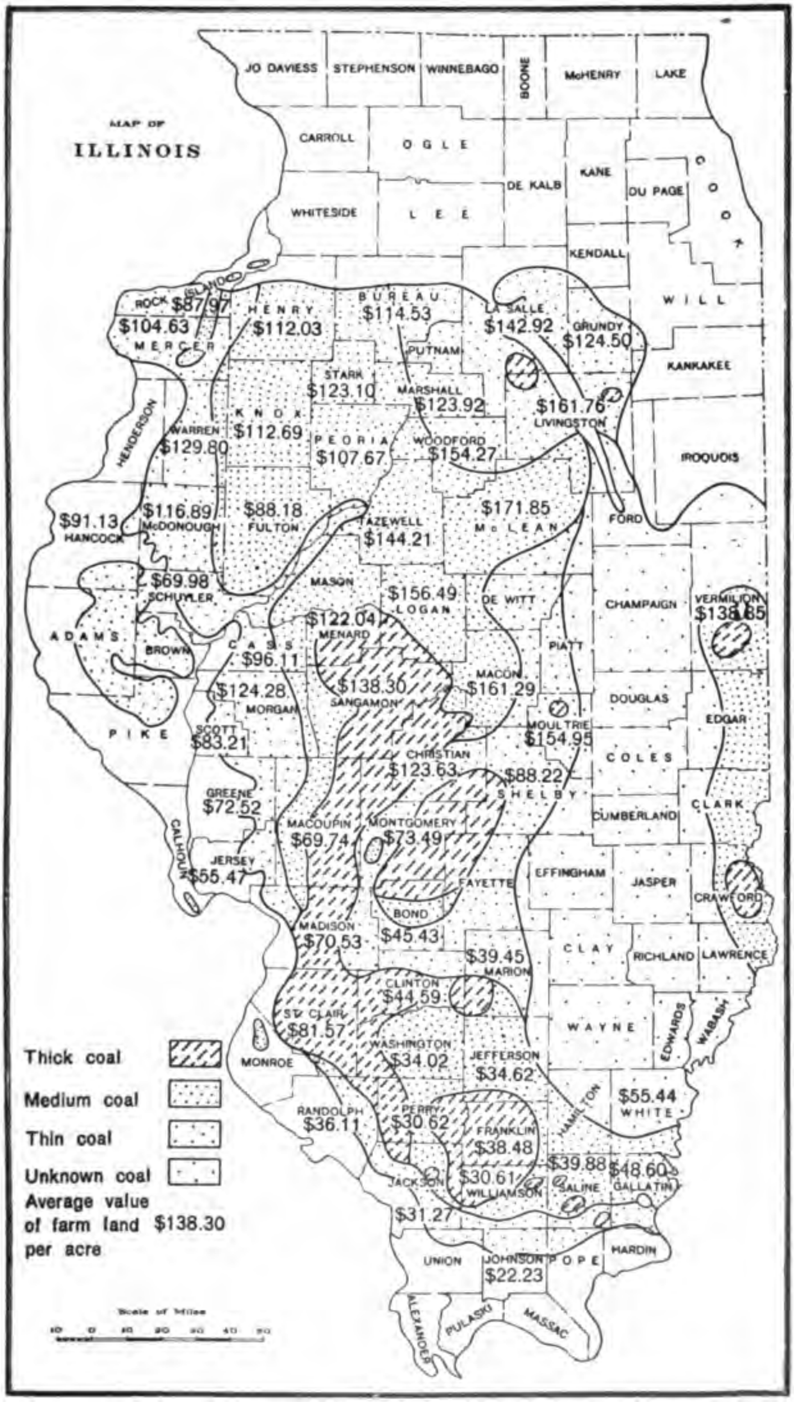

FIG. 1. MAP SHOWING THICKNESSES OF COAL AND VALUES OF FARM LANDS, AS GIVEN BY THE 1910 CENSUS REPORTS

paired, especially if it were accepted that mining is certain to result in subsidence and operations were so planned as to reduce the surface damage and the cost of restoration to the lowest possible amount.

When the coal producer owns nothing but the coal rights, unreasonable damages for surface subsidence are sometimes imposed. The measure of the damages is not always merely the decreased productive value of the land, nor the cost of restoring it to its former condition by artificial drainage; the formation of a small pond has often been claimed to lower the market value of a farm to a considerable extent, simply because it made the farm less sightly. Under these circumstances, operators naturally desire to avoid disturbing the surface for they know that an attempt will be made to recover damages and that, even if they escape the payment of exorbitant amounts, they will incur considerable expense in defending the suits. An effort, accordingly, is often made to conduct the mining operations in a manner which will not result in surface subsidence. The result is that the loss in the ground represents an important percentage of the coal, in many cases more than half that contained in the area worked.

It is very important that the allowable damages for surface subsidence be regulated by some law. This law should fix the damage payable by the coal producer, in case he is legally responsible, upon the basis of the actual damage done to the surface. Under such a law the operators would know the extent and character of their responsibility and could, without fear of excessive or unreasonable damages, proceed according to methods which would yield the highest possible percentage of extraction justifiable under such conditions.

7. *Squeezes.*—Closely related to, but not identical with, the subject of surface subsidence is that of squeezes. There may be subsidence without a squeeze, but with the conditions in Illinois a squeeze is usually followed by subsidence. The removal of a portion of a deposit throws additional weight upon the pillars left and if these pillars are not strong enough to support this additional weight, they will crack and crush, causing a movement of the overlying material. This movement is called a "squeeze," or sometimes a "creep." Large quantities of coal are often left in the mine in the form of pillars in an effort to prevent a squeeze, which may not only interfere with the operation of the mine and entail a loss of

the coal in the squeezed area, but large areas of the mine may become inaccessible for future economical working. A more nearly complete extraction of the coal properly carried out should, however, result in less damage from squeezes.

There are two ways in which a squeeze may be prevented or stopped; first, by the use of a support strong enough to prevent any movement of the overlying rock, and secondly, by a fracture of the rock above the excavated portion so that the weight on the pillars will not be sufficient to crush them.

The first method may be employed by either leaving natural supports (coal pillars) of sufficient size and strength to hold the roof without any movement, or by the use of artificial supports, such as timber or iron columns, or sand or culm filling. The cost of timber or iron would prohibit their use if a large percentage of coal was to be extracted. Filling is not generally non-compressible, but it occupies most of the space from which coal has been removed, prevents any scaling off of pillars, and eliminates any possibility of movement of pillars. The filling method, however, is hardly to be considered feasible in Illinois because of the cost. In the Upper Silesian coal field where this method is most extensively used, the cost is from twelve to eighteen cents per ton.*

Another difficulty arises from the fact that the material would have to be flushed into the mine with water, and then the water would have to be pumped out. This water would probably have an injurious effect on the clay bottom. The material, moreover, would have to be brought from a distance unless the value of the land should be so small as to permit the use of material from the neighboring surface, and this condition would rarely prevail in Illinois.

In the leaving of coal pillars of sufficient strength to prevent roof movement, the amount of coal which must be left varies with local conditions. It is difficult, if not impossible, to determine this factor in advance, and in attempting to approach as closely as possible the limit of safety, it often occurs that too much coal is removed. Even if the limit is not passed so far as immediate movement is concerned, it may be passed with reference to ultimate movement and the crushing of the pillars.

Apparently, so far as a large part of the State of Illinois is concerned, it is necessary to leave in the ground about one-half of the area of the coal if movement of the overlying beds is to be prevented.

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*Gullachsen, Berent Conrad, "The Working of the Thick Coal Seams in Upper Silesia," Trans. Inst. Min. Engrs. Vol. 42, p. 209, 1911.

The desirable dimensions of the rooms and the pillars vary widely from wide pillars between wide rooms to narrow pillars between narrow rooms. One company had squeezes when it drove 25-foot rooms on 50-foot centers, but it has had no trouble with 30-foot rooms with 60-foot centers. This is a question not simply of the crushing strength of the coal nor of the ability of the bottom to withstand pressure, but of the effect on the pillars of scaling at the sides. In other words, the strength of the pillar is not determined merely by its original size but by its effective size after the scaling action, which may follow the extraction of the room coal, has occurred. This scaling action is increased by the shattering effect of explosives.

The use of coal in the ground to prevent squeezes and subsidence, which is what abandoning of pillar coal amounts to, ought to be considered only as a last resort. It has been found that squeezes can be prevented by the removal of so little coal on the advance as to leave a solid support, and by the complete removal on the retreat so that the roof is left entirely without support. This process prevents the gradual settling which occurs when some support is left and produces a sharp bending, or localization of stress, sufficient to cause a rupture of the overlying rock and prevent the transference of weight from the mined-out area to the standing coal. This is the only certain method which has been found for the prevention of squeezes unless an absurdly large quantity of coal is abandoned. The means by which squeezes may be prevented vary under different conditions, but the essential consideration is that the roof of the mined-out area shall be left absolutely without support either from coal or from timber, so that it must fall.

TABLE 1[1]

DIMENSIONS OF ROOMS AND OF ROOM PILLARS AND PERCENTAGE OF EXTRACTION

LOCALITY AND COAL SEAM	Width of Room Feet	Width of Room Pillars Feet	Per Cent of Coal Left in First Working	Total Per Cent of Coal Recovered
Alabama:				
Newcastle seam	25	10–20	40	85
Thin seam	20	10–20	30	90
Blue Creek	25	25	35	85
Blocton	25	25	30	80
Flat Top	25	25	35	85
Arkansas:				
Sebastian County	18–30, usually 24	12	30	70
Colorado:				
Trinidad series	20–25	20–25	50	50–80
Illinois:				
Springfield	30	15–20	60	60
Staunton (machine mines)	30	30	60	60
Indian Territory	18–30, usually 24	10–12	30	70
Iowa	21–30	9–15	35	90
Maryland:				
Georges Creek	12–16	40–100	75	90
Pennsylvania:				
Connellsville	12	42	60	95
Connellsville	12	84	80	90
Pittsburgh	21–24	12–18	40	85–95
Clearfield	21–24	15	40–50	80–95
West Virginia:				
Fairmont	18–24	18–40	40–60	95
Clarksburg, Pittsburg seam	18	32	64	90
Mineral County, Pittsburg seam	12	48	80	95 (?)
Tucker County, Davis or Kittanning seam	18	22	35	90 (?)
Putnam County, Pittsburg seam	30	15	34	75–85
New River, Sewell-Nuttall seam	24	26	60	60–80
Thacker, (Thacker) Lower Kittanning seam	18–21	42	67	80
Logan, Lower Kittanning seam	21	20	57	....
Kanawha (No. 2 Gas), Lower Kittanning seam	28	12–20	40	....
Pocahontas, Double Entry, No. 3 Pocahontas seam	18	42	68	80
Pocahontas, Panel System, No. 3 Pocahontas seam	18	42	....	95[2]
Raleigh, Sewell-Nuttall seam	25	25	50	90

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[1] H. H. Stoek, Mines and Minerals, Vol. 26, p. 107, 1905, and International Library of Technology, Vol. 150, par. 40, p. 60.

[2] Estimated.

The widths of rooms and room-pillars given in Table 1 show that there were few cases in which plans were made for the extraction of more coal from pillars than from the advance workings.

TABLE 2[1]

PRINCIPAL FACTORS GOVERNING RECOVERY OF COAL IN DIFFERENT DISTRICTS

Item	Operating District and State	Per Cent of Recovery Entire Seam	Ultimate Recovery Compared to Present	Period of Operation Years	Average Height of Seam, Feet	Roof Coal Carried	Nature of Top	Nature of Bottom	Pillar-Drawings	Clay Veins Encountered
1	Colorado, Southern	80–90	Same	5–30	8.5	24 in.	Slate	Soft slate	Yes	Dikes
2	Colorado, Other districts	60–65	Same	5–30	8.5	18–24 in.	Very soft	Soft slate	Yes	None
3	Colorado, Other districts	75–80	Better	10–35	3–7	Few places	Sandstone, poor shale	Same as top	Yes	None
4	Michigan, Saginaw District	65 80–90	Better	15	3	None	Black slate	Fire clay	Where allowed	None
5	Illinois, Central	50	Same	20	8	None	Slate, clod, and limestone	Fire clay	None	None
6	Illinois, Southern	65–70	Same	20	8	Yes	Sandy shale	Fire clay	To some extent	None
7	Illinois, Springfield District	55–75	Increase[2]	20–25	6–7	Some places	Hard shale	Fire clay	Where allowed	None
8	Illinois, Williamson and Saline Counties	55–75	Increase[2]	18	5.5	Some places	Hard shale	Fire clay	Where allowed	None
9	Illinois, Sherrard Field	90	Same	20	3.7	None	Blue rock and cap rock	Slate and sand rock	Yes	Yes
10	Pennsylvania, Extreme southwest section	72.5	Better	3	7.5	10 in.	1.5–3 ft. draw slate	Soft fire clay	Yes	None
11	Pennsylvania, Westmoreland Co.	84	Below	25–35	6.7	None	Slate	Fire clay	Yes	None
12	Pennsylvania, Somerset County	95	Same	8	3.9	None	Hard black slate	Limestone	Yes	Very few
13	Maryland, Georges Creek Field	97	Same	12	3	None	Sand rock	Sand rock	Yes	Yes
14	Maryland, Georges Creek Field	88	Same	94	9	18 in.	Gray shale coal, dark shale	Hard gray shale	Yes	None
15	Ohio, Belmont County	60	Same	....	5–6	None	Slate and shale	Fire clay	None	None
16	Ohio, E. Harrison County	70–75	Same	....	3.7–5	None	10 in. firm slate	Fire clay	None	None
17	West Virginia	90	Same	50	8	Some places	Varies	Fire clay	Yes	Yes
18	Alabama, no reply									
19	Tennessee, no reply									
20	Kansas, no reply									
21	Iowa, no reply									
22	Kentucky, no reply									

[1] A. W. Hesse, "Maximum Coal Recovery," W. Va. Min. Inst., June 3, 1914; Coll. Eng., Vol. 35, p. 13; and Coal Age, Vol. 5, p. 1051.

[2] Operated by panel method, all others by room-and-pillar.

TABLE 3

DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Coöp. Investigations)	Depth of hoisting shaft	Method		Entry width			Entry pillar width			Barrier pillar width		No. rooms on room entry	Room		Width of room pillar	Room neck		Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezes?	Operators' Estimate of Percentage Extracted
				Unmodified Room-and-Pillar	Panel	Main	Cross	Room	Main	Cross	Room	Main	Cross		Width	Length		Width	Length						
II	2	12	125	X	—	8	8	—	12	12	—	23	—	—	24	150	18	8	10	20	42	34	8	No	44
	2	13	114	—	X	9	9	9	20	20	20	30	30	16	24	250	20	9	4	20	44	35	9	No	55
	2	14	135	X	—	8	8	—	18	18	—	30	—	—	28	250	22	18	12	22	50	42	8	No	46
	—	15	160	X	—	8	8	—	18	18	—	20	—	—	25	240	17	8	10	20	42	34	10	No	49
III	1	17	210	X	—	8	8	—	30	30	—	50	—	—	20	250	20	8	7	14	40	32	12	No	96
	1	18	69	X	—	12	12	—	42	15	—	36	—	—	26	250	19	8	7	12	45	37	12	Yes	96
	1	19	70	X	—	12	12	—	18	18	—	18	—	—	24	250	18	8	7	25	42	34	8	No	50
	2	22	60	X	—	6	6	—	..	..	—	..	—	—	15	250	15	6	15	25	30	24	15	Yes	45
	1	24	40	X	—	12	12	—	20	20	—	15	—	—	24	300	21	8	7	29	45	37	15	No	70
IV	5	26	196	X	—	8	8	—	30	21	—	50	—	—	26	190	6	8	8	30	32	24	8	No	56
		29	185	X	—	8	8	—	36	24	—	40	—	—	24	180	8	8	7	18	32	24	8	No	57
		30	60	X	—	10	8	—	15	12	—	25	—	—	22	210	10	8	7	18	32	24	13	No	57
		34	200	X	—	8	10	—	30	34	—	38	—	—	28	180	10	10	9	30	38	28	4	Yes	42
		41	365	X	—	16	16	—	20	30	—	80	—	—	25	200	10	12	9	15	35	23	8	No	56
		25	185	—	X	8	8	8	36	30	24	50	24	23	24	180	6	8	7	18	30	22	8	Yes	54
		27	170	—	X	12	12	12	20	15	12	50	50	30	30	200	8	8	9	20	38	30	8	No	53
		28	150	—	X	8	8	8	25	20	8	40	20	18	24	180	10	8	10	15	34	26	8	Yes	54
		32	68	—	X	8	8	8	30	30	24	20	20	30	20	150	12	8	10	15	32	24	8	No	57
		33	285	—	X	8	8	8	20	20	20	50	40	15	30	150	6	8	10	35	36	28	6	No	64
		35	187	—	X	8	8	8	35	35	30	60	50	14	26	200	9	8	9	27	35	27	8	Yes	56
		36	238	—	X	12	12	12	30	24	10	30	20	24	25	240	10	8	9	27	35	27	12	Yes	48
		37	235	—	X	10	10	10	30	30	20	60	40	18	30	240	12	12	15	15	42	30	10	Yes	41
		38	245	—	X	10	10	10	27	24	20	35	35	25	30	200	6	10	9	20	36	26	10	No	62
		39	204	—	X	12	12	12	30	30	20	30	30	15	30	250	12	12	11	18	42	30	10	Yes	65
		40	270	—	X	10	10	10	30	24	20	35	35	27	24	200	6	8	9	18	30	22	7	Yes	50
V	5	43	160	X	—	14	14	—	30	30	—	40	—	—	30	250	20	18	10	20	50	32	18	Yes	65
		44	270	X	—	15	18	—	25	25	—	50	—	—	22	250	24	18	12	18	46	28	18	Yes	59
		45	320	X	—	16	16	—	24	20	—	..	—	—	27	250	20	19	12	30	47	28	15	No	64
		46	450	X	—	14	14	—	20	20	—	50	—	—	33	200	7	19	12	25	40	21	19	Yes	82
		47	90	X	—	14	19	—	20	12	—	20	—	—	22	250	12	8	6	13	34	26	12	No	68
		48	76½	X	—	14	14	—	25	25	—	50	—	—	24	300	21	12	6	15	45	33	12	No	59
		49	337	X	—	14	14	—	30	21	—	35	—	—	24	250	9	12	9	25	33	21	12	Yes	73

TABLE 3 (Continued)

DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Coöp. Investigations)	Depth of hoisting shaft	Method		Entry width			Entry Pillar width			Barrier Pillar width		No. rooms on room entry	Room		Width of room pillar	Room neck		Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezes?	Operators' Estimate of Percentage Extracted
				Unmodified Room-and-Pillar	Panel	Main	Cross	Room	Main	Cross	Room	Main	Cross		Width	Length		Width	Length						
VI	6	50	726	×	—	12	12	—	40	100	—	35	—	—	22	250	25	12	24	34	47	35	22	Yes	58
		55	160	×	—	10	9	—	50	50	—	50	—	—	25	250	26	15	12	19	51	36	15	Yes	46
		56	580	×	—	12	12	—	30	100	—	40	—	—	26	250	26	12	23	45	52	40	11	No	50
		59	200	×	—	9	9	—	20	50	—	20	—	—	23	225	15	12	16	32	38	26	15	Yes	68
		60	220	×	—	12	12	—	18	20	—	20	—	—	21	300	12	12	6	24	33	21	12	Yes	51
		62	190	×	—	10	10	—	30	75	—	30	—	—	20	250	15	10	15	15	35	25	10	Yes	51
		63	140	×	—	10	10	—	30	40	—	30	—	—	22	500	18	10	12	18	40	30	12	Yes	..
		64	120	×	—	11	11	—	19	38	—	22	—	—	20	300	15	9	8	15	35	26	8	No	64
		65	90	×	—	12	12	—	20	10	—	20	—	—	21	250	9	10	6	24	30	20	10	Yes	58
		51	494	—	×	9	9	9	30	30	30	100	100	16	24	250	16	9	18	38	40	31	18	Yes	52
		52	640	—	×	12	12	12	50	50	38	150	100	30	25	250	25	12	18	41	50	38	12	No	58
		53	450	—	×	12	16	16	28	50	50	100	100	35	29	300	11	20	18	18	40	22	..	Yes	65
		54	380	—	×	15	7	7	50	50	30	150	150	21	20	250	25	9	11	35	45	34	7	Yes	58
		57	320	—	×	12	12	12	50	38	38	100	100	20	25	300	25	12	10	32	50	38	18	No	55
		58	516	—	×	12	12	12	25	20	20	125	125	32	22	200	12	12	15	34	34	22	12	Yes	56
VII	6	68	387	×	—	21	16	—	40	40	—	27	—	—	32	235	33	9	15	27	65	56	20	Yes	50
		70	92	×	—	14	14	—	50	50	—	50	—	—	30	300	30	21	25	50	60	39	21	No	50
		72	287	×	—	21	21	—	40	40	—	..	—	—	30	300	30	21	6	..	60	39	21	No	56
		78	160	×	—	21	21	—	55	45	—	55	—	—	32	250	41	21	24	34	73	52	21	Yes	56
		79	127	×	—	21	21	—	60	30	—	60	—	—	30	300	30	21	12	24	60	39	21	No	62
		80	145	×	—	12	14	—	18	35	—	60	—	—	30	250	30	21	18	38	60	39	21	No	59
		81	200	×	—	21	21	—	80	40	—	60	—	—	35	250	45	21	25	40	80	59	21	Yes	55
		82	192	×	—	14	21	—	42	42	—	60	—	—	40	250	40	21	18	36	80	59	30	No	63
		84	320	×	—	21	21	—	39	39	—	40	—	—	30	240	30	21	15	25	60	39	21	No	60
		87	707	×	—	12	12	—	40	40	—	150	—	—	30	300	30	12	20	50	60	48	20	No	58
		88	85	×	—	12	12	—	20	20	—	40	—	—	29	250	17	12	6	16	46	34	12	Yes	65
		89	85	×	—	8	8	—	20	15	—	40	—	—	25	200	20	8	10	24	45	37	8	Yes	48
		90	160	×	—	21	9	—	60	30	—	30	—	—	33	250	17	9	12	24	50	41	12	No	63
		66	832	—	×	21	21	21	40	40	40	110	75	30	30	300	30	21	25	50	60	39	21	Yes	56
		67	320	—	×	12	12	12	30	30	24	60	60	30	30	300	10	9	6	32	40	31	9	No	63
		69	290	—	×	12	12	12	30	30	30	60	60	27	23	200	22	12	9	30	45	33	12	No	47
		71	194	—	×	10	10	10	40	40	40	75	60	..	38	200	27	9	18	36	65	56	10	Yes	51
		73	318	—	×	21	21	21	60	40	13	60	60	13	30	300	20	21	12	30	50	29	21	No	60
		74	330	—	×	12	12	12	60	60	12	50	50	14	30	265	30	12	12	30	60	48	12	Yes	50
		75	310	—	×	12	12	12	50	40	25	50	50	25	30	250	18	12	18	30	48	36	12	Yes	52
		76	870	—	×	10	10	10	50	40	..	75	75	..	28	225	27	10	20	35	55	45	10	Yes	44
		77	462	—	×	14	14	14	50	50	40	100	100	40	30	250	25	22	22	28	55	33	21	Yes	63
		83	140	—	×	21	21	21	35	35	12	35	35	12	27	600	43	21	12	24	70	49	21	Yes	50
		85	440	—	×	12	12	12	50	50	25	50	50	25	35	250	19	9	9	25	44	35	12	Yes	55
		86	536	—	×	12	8	12	100	50	16	100	50	16	26	400	25	12	18	30	50	32	8	No	52

TABLE 3 (Concluded)

DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Coöp. Investigations)	Depth of hoisting shaft	Method		Entry width			Entry Pillar width			Barrier Pillar width		No. rooms on room entry	Room		Width of room pillar	Room neck		Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezes?	Operators' Estimate of Percentage Extracted
				Unmodified Room-and-Pillar	Panel	Main	Cross	Room	Main	Cross	Room	Main	Cross		Width	Length		Width	Length						
VIII	6	91	217	×	—	9	9	—	25	21	—	40	—	—	43	200	4	9	19	18	47	38	5	Yes	55
	6	92	240	×	—	7	7	—	35	30	—	60	—	—	24	240	3	9	18	18	27	18	9	Yes	68
	6	93	186	×	—	8	9	—	21	21	—	50	—	—	24	200	16	9	9	9	40	31	9	Yes	75
	7	94	90	×	—	7	6	—	16	12	—	17	—	—	24	240	6	9	9	9	30	21	9	No	81
	6	95	90	×	—	6	6	—	..	..	—	..	—	—	25	500	11	9	12	12	36	27	9	No	82
	7	97	223	×	—	7	8	—	21	21	—	50	—	—	21	150	9	9	12	12	30	21	7	No	68
Averages by districts II	2		133	—	—	8	8	—	17	17	—	25	—	—	25	222	19	8	9	21	45	36	9	—	48
III	1 & 2		90	×	—	10	10	—	28	21	—	30	—	—	22	260	18	8	9	21	40	33	12	—	71
IV	5		202	—	—	10	10	—	27	25	—	44	—	—	26	197	9	9	9	21	35	26	9	—	55
V	5		243	×	—	15	16	—	25	22	—	41	—	—	26	250	16	15	10	21	42	27	15	—	67
VI	6		348	—	—	11	11	—	33	33	—	81	—	—	23	262	18	11	14	28	41	30	13	—	56
VII	6		279	—	—	16	15	—	46	39	—	60	—	—	31	277	28	16	16	32	58	42	17	—	56
VIII	6 & 7		174	×	—	7	7	—	24	21	—	43	—	—	27	255	8	9	13	13	35	26	8	—	71
Aver. of 30 panel mines			306	—	×	12	12	12	39	35	25	68	56	23	27	251	18	12	13	24	45	33	11	...	55
Aver. of 48 room-and-pillar-mines.........			208	×	—	12	12	..	31	27	..	46	..	..	26	250	19	12	12	24	45	33	13	...	54

TABLE 4[1]

VALUES[2] OF SURFACE AND OF COAL RIGHTS BY COUNTIES IN ILLINOIS

County	Value of Coal per Acre	Number of Coal Bed	Average Surface Value, Census of 1910
Bond	$ 25	6	$ 45.43
Bureau	10–100	2	114.53
Christian	10– 50	6	123.63
Franklin	35–100	6	38.48
Fulton	15–100	5	88.18
Gallatin	20– 25	5	48.60
Grundy	10– 25	2	75.52
Henry	135	6	112.03
Jackson	25– 75	2, 6	31.27
La Salle	10–100	2, 5	142.92
Livingston	10– 50	6	161.76
Logan	20– 50	5	156.49
Macoupin	15– 50	6	69.74
Madison	10– 40	6	70.53
Marion	20	6	39.45
Marshall	15	2	123.92
McLean	15	5	171.85
Menard	25– 30	6	122.04
Montgomery	25– 50	6	73.49
Morgan	20– 30	6	124.28
Peoria	20– 50	5	107.67
Perry	25	6	30.62
Putnam	15	2	104.69
Randolph	25	6	86.11
St. Clair	10–100	6	81.57
Saline	50–150	5	39.88
Sangamon	20–100	5, 6	138.30
Scott	10– 40	2	83.21
Shelby	10– 25	6, 5	88.72
Vermilion	100–150	6, 7	138.85
Warren	15	1, 2	129.80
Will	15	2	104.08
Williamson	50–150	6	30.61
Woodford	15	2	154.27

[1] Young, Lewis E., "Surface Subsidence in Illinois," Ill. Coal Min. Invest., Bulletin 17, p. 55, 1916.

[2] These prices are not offered as an authoritative basis for valuation but indicate in a general manner the prices at which coal has been sold or at which it is held in some of the important counties.

CHAPTER II

EXTRACTION IN ILLINOIS

8. *Plan for Division of State into Districts.*—At the beginning of the work of the Coöperative Coal Mining Investigations, the State was divided into districts in order that those beds which are similar in general conditions might be studied and considered together. This subdivision into districts is shown by Fig. 2, and the districts are described in Table 5.

TABLE 5

DISTRICTS INTO WHICH THE STATE HAS BEEN DIVIDED FOR THE PURPOSES OF INVESTIGATION

Investigations, District	Coal Seam	Method of Mining	Counties	Investigations Numbers for Mines Examined
I	2	Long-wall	Bureau, Grundy, La Salle, Marshall, Putnam, Will, Woodford	1 to 11
II	2	Room-and-pillar	Jackson	12 to 16
III	1 and 2	Room-and-pillar	Brown, Calhoun, Cass, Fulton, Greene, Hancock, Henry, Jersey, Knox, McDonough, Mercer, Morgan, Rock Island, Schuyler, Scott, Warren	17 to 24
IV	5	Room-and-pillar	Cass, DeWitt, Fulton, Knox, Logan, Macon, Mason, McLean, Menard, Peoria, Sangamon, Schuyler, Tasewell, Woodford	25 to 42
V	5	Room-and-pillar	Gallatin, Saline	43 to 49
VI	6 (east of Duquoin anticline)	Room-and-pillar	Franklin, Jackson, Perry, Williamson	50 to 65
VII	6 (west of Duqoin anticline)	Room-and-pillar	Bond, Christian, Clinton, Macoupin, Madison, Marion, Montgomery, Moultrie, Perry, Randolph, Sangamon, Shelby, St. Clair, Washington	66 to 90
VIII	6 and 7 (Danville)	Room-and-pillar	Edgar, Vermilion	91 to 97

In the present publication the conditions prevailing and the methods followed in the various districts are described, and the extent to which these affect the percentage of recovery is discussed. Material and information has been gathered at various times, and

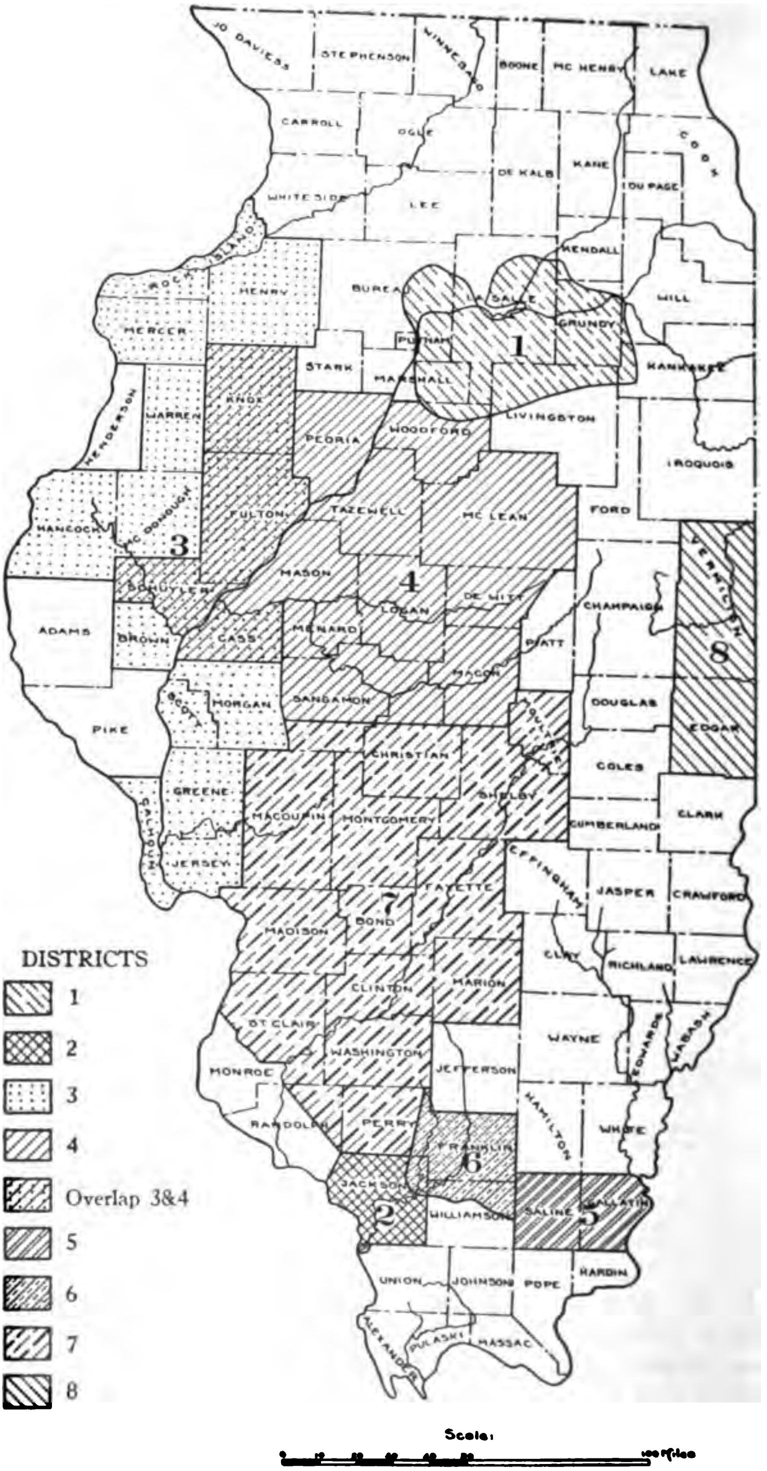

Fig. 2. Map of Districts of Coöperative Coal Mining Investigations

some of it, especially that relating to physical conditions and usual methods of operation, has been published in previous bulletins of this series. These facts are summarized in Bulletin 13.*

9. *Conditions Affecting Extraction.*—Since there is an immense quantity of coal underlying the state and only a comparatively small portion has been extracted, it is perhaps natural that little serious attention has been given to the subject of high recovery. Those controlling production have been concerned principally with other phases of the subject, not because they have been indifferent to the highest possible utilization of resources, but because they have believed that the methods in use were giving the lowest possible cost of production; and low cost of production has been regarded as a necessity for the development of the Illinois fields in competition with other coal fields.

Table 3, rearranged from Bulletin 13, gives the dimensions of the workings and the estimates of recovery for the mines examined by the Coöperative Coal Mining Investigations.

The values for the percentage of extraction given in the last column of Table 3 are, in most cases, founded upon estimates furnished by the operators. In many instances subsequent investigation has shown that these values are not correct. There are only a few mines in the state from which it has been possible to obtain accurate data on recovery because of the lack of information on which such data could be based. Generally, it has been found that persons estimating the percentage of recovery have been inclined to use values too high and have failed to take into account some of the sources of loss. Later figures on extraction, the most trustworthy it has been possible to obtain, will be found in the descriptions of the districts.

10. *District I.*—The No. 2 bed varies in thickness from two feet, eight inches to four feet, the average thickness being about three feet, two inches. On the east side of the LaSalle anticline the thickness of cover ranges from 40 to 200 feet; on the west side the bed lies at a depth of 350 to 550 feet. In the eastern, or Wilmington, section the roof is a smooth gray shale, though sandstone is found in some places. In the western or LaSalle field the roof is a gray

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* Andros, S. O., "Coal Mining in Illinois," Ill. Coal Min. Invest., Bul. 13, 1915.

shale. In the Wilmington field the floor is a dark gray fire clay varying in thickness from a few inches to several feet. When this clay is wet, it heaves badly under pressure. In the LaSalle field the floor is fire clay, but a hard sandstone is sometimes found immediately beneath the coal.*

Nearly all the coal produced in this district is mined by the long-wall method, and this method, of course, gives the highest possible percentage of recovery. G. S. Rice says that at one mine in which a record was kept for six years the loss of coal from all causes was five per cent.†

11. *District II.*—The No. 2 seam is found under shallow cover ranging from 25 to 160 feet. In most places the floor is sandstone, but shale or clay is occasionally found. In places a wet and fluid sand is found about thirty feet below the surface, and it has a marked effect upon surface subsidence, causing the formation of rather deep pits instead of gentle sags. The bed is divided into two benches by a shale parting, varying in thickness from one-eighth inch to thirty-six feet. The bottom bench varies in thickness from 3½ to 4 feet, and the top bench has an average thickness of two feet. Where the parting between the benches is less than four inches thick, the two benches of the seam are worked as one and the working faces in rooms and entries are from six to seven feet high. Where the parting is more than four inches thick, only the lower bench is mined and the parting becomes the mine roof. When both benches are worked and the bed is more than six feet thick, only the lower six feet of coal are mined, eight to twelve inches of top coal being left; but if the coal is not more than six feet thick the full thickness of the bed is mined, and the gray shale overlying the coal becomes the roof.

With one exception, the mines examined are operated by the unmodified room-and-pillar method. Operations are carried on without close adherence to the projected sizes of rooms and pillars. The result of this practice is a rather high percentage of extraction, as pillars are gouged to a considerable extent.‡ At one mine in this district which is operated on the panel system and in which a serious attempt is made to remove pillars as far as possible, the percentage of extraction is probably higher than at any other mine in southern Illinois. At this mine the shaft is 115 feet deep. There are triple main and cross entries, each ten feet wide, with 20-foot entry pillars. Barrier pillars on main and cross entries are twenty feet wide. Rooms are twenty feet wide with 10-foot pillars. All cross-cuts are eight feet wide. Although there are no exact figures on the percentage of recovery, it is evident from the dimensions of the workings that about two-thirds of the coal is extracted in the first working. Since by slabbing pillars, forty to fifty per cent of the pillar coal is also obtained, the final recovery probably amounts to about eighty per cent. The rooms are widened about thirty feet before the end is reached, little or no pillar coal is left beyond this point, and as much of the remainder of the pillars as possible is taken out by slabbing.

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*Andros, S. O., "Coal Mining Practice in District I," Ill. Coal Min. Invest., Bul. 5, p. 10, 1914.

†Rice, George S., "Mining-Wastes and Mining-Costs in Illinois," Trans. Amer. Inst. Min. Engrs., Vol. 40, p. 31, 1909.

‡Andros, S. O., "Coal Mining Practice in District II," Ill. Coal Min. Invest., Bul. 7, p. 9, 1914.

The possibility of extracting a large amount of pillar coal depends upon the character of the top which may be allowed to fall without serious consequences, because the shale and sand overlying the coal seal the opening so that the influx of water is not seriously increased by a break. When the top falls, the necks of the rooms are boarded up and the water is handled by a pump.

12. *District III.*—The No. 1 and No. 2 beds are worked. The cover overlying the coal is thin. The topography of the surface in many places is rolling, with hills about 150 feet high near Matherville. Bed No. 2 lies at depths of seven feet to one hundred feet with an average cover of fifty-five feet. Bed No. 1 averages four feet in thickness and is broken in places by small faults, slips, clay veins, and rolls. A poorly developed parting divides the bed into two benches, the upper of which is in most places about two feet thick.

The immediate roof in the northwestern part of the district is of hard black shale which is easy to support. In the southern part a bituminous calcareous shale, two to five inches thick, lies in places immediately over the coal. This shale, called clod, is hard when first exposed to the air but after exposure softens and falls. Throughout the district the cap rock is limestone. In limited areas where the shale is missing, this limestone forms the immediate roof. Above the cap rock occurs a dense, fine-grained, non-crystalline limestone locally called " blue rock."

Below bed No. 1 there occurs in places an irregular band of hard bone, three to six inches thick. The floor proper is of light gray micaceous fire clay which contains plant stems and roots. This clay heaves badly when wet and sometimes swells enough to fill the entry. In parts of some mines a carbonaceous shale lies between the fire clay floor and the coal; sometimes this shale is supplanted by sandstone. These casual deposits are called " false bottoms."

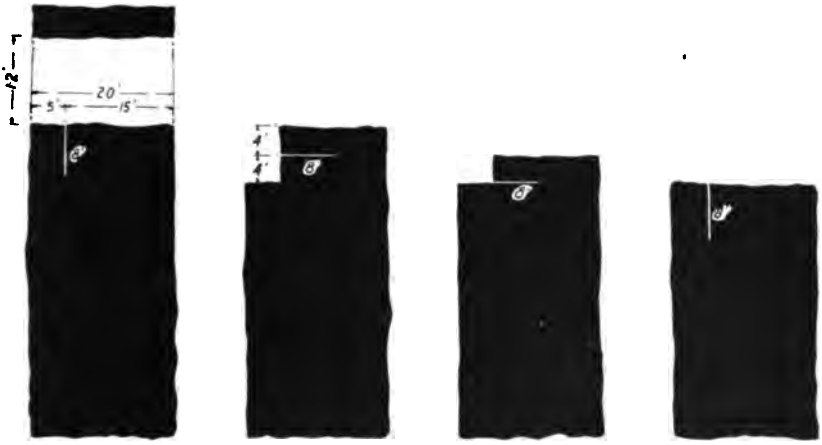
Bed No. 2 varies in thickness from 1 foot, 10 inches to 4 feet, and averages 2 feet, 6 inches. The bed has a slight dip to the east. A band of mother coal and iron pyrites persists throughout the bed. This occurs about fourteen inches from the roof. The immediate roof is of smooth and regular calcareous shale, known locally as soapstone. The floor is of soft gray fire clay which contains nodular concretions of iron pyrites called sulphur balls. The coal in this district lies near the surface, but at no point is the overburden stripped.

Except at two mines, the mining system is the simplest form of double-entry room-and-pillar. Table 3 shows the dimensions of workings in the mines examined. The coal is gained during the first working with a waste of pillar coal amounting to about 45 per cent of the bed. At the two exceptions 75 per cent of the pillar coal is recovered on the retreat, a large percentage for Illinois room-and-pillar mines.

A main entry and a parallel air-course, each six feet high and eight feet wide, are driven from each side of the shaft toward the boundaries. At right angles to these main entries, pairs of cross entries are driven every 500 feet. On the cross entries, after leaving a barrier pillar of 50 feet, rooms are turned on 45-foot centers. Room necks are 7 feet long and 8 feet wide, and are widened to the left at angles of about 45 degrees; thus they reach the full room width of 26 feet at distance of 14 feet from the beginning of the widening. After the first room on each entry has been holed through, the room-pillar cross-cuts are closed by gob stoppings, and the line of No. 1 rooms is kept open; thus two additional air-courses are provided.

After the entry has been driven to the limit and the rooms on it have been worked out, the last pillar on the entry is drawn; then the other room pillars are drawn until the pillar between rooms

3 and 4 is reached. The room pillars between the main entry and room 4 are left to protect the main entry and air-course. The method of drawing pillars is illustrated in Fig. 3. When the room is driven up to its full length, a 12-foot cut is made across the end of the pillar (a), a 5-foot slab about 8 feet long is shot from the side of the pillar, a 4-foot slab is shot from the end (b), and the end of the pillar is squared up by shooting off another 4-foot slab (c). Beginning again at (d), the process is repeated.



(a) (b) (c) (d)

FIG. 3. PILLAR DRAWING AT MATHERVILLE, ILLINOIS

The hard roof is easy to support and often stands while 25 to 200 feet of pillars are being drawn. When the weight of the roof becomes too heavy, the roof breaks at the pillar ends. The cracking of the props gives ample warning of the break, and work is discontinued until the roof falls. The interval between the first heavy cracking of props and the breaking of the roof is usually not more than twelve hours.

A break line of about twenty-five degrees with the face of the rooms is roughly maintained. When roof falls prevent access to the squared-up pillar ends, a 12-foot cut is again made completely through the pillar, as at the face of the room when drawing began, and with this new pillar end the procedure continues; consequently, very little pillar coal is lost. Carl Scholz, President of the Coal

Valley Mining Company, states that at mine No. 3 at Matherville the loss of pillar coal does not exceed four per cent.

At the No. 3 mine of the Coal Valley Mining Company, the cost of producing coal is much less on pillars than on advance work in rooms. Room coal costs on the average $1.25 per ton at the pit mouth, and pillar coal costs $1.015. This difference in cost exists because track, yardage, bottom digging, and driving through rolls and slips are properly charged against room coal, while there are no such charges against pillar coal. When pillars are drawn, therefore, the average cost per ton for the total production is materially reduced. At this mine rooms are worked with one man at the face, but two men are placed at each pillar and at the face of each entry. Only one man has been injured in connection with the pillar drawing.

With the extraction of such a large percentage of the bed surface subsidence is to be expected. The topography of the surface is rolling, and subsidence is usually indicated by cracks in the hillsides. The largest single area affected was reported to be one acre which subsided from 6 to 12 inches.*

13. *District IV.*—In District IV the No. 5 coal is mined. The average thickness of this coal is 4 feet, 8 inches according to data taken at 240 mines and given in the Thirty-first Annual Coal Report of Illinois. The No. 5 bed outcrops in Peoria, Fulton, and Knox Counties, but is found at greater depths toward the east. It lies from 300 to 600 feet below the surface in Macon County, 400 feet in McLean, and from 260 to 300 feet in Logan.

The roof is of black sheety shale varying in thickness from a few inches to 35 feet and containing occasionally "niggerheads" of pyrite. In many mines there is, in places, a layer of pyrite two or three inches thick between the coal and the shale. Where this layer is present, the shale is protected from the air and stays up; where it is not present, the shale falls badly and sometimes caves to a height of 35 feet. A limestone occurs above the shale in most mines, though in a few places a fine grained micaceous sandstone is found. In some cases the shale is absent, and the cap rock becomes the roof.

A great many clay veins extend through the coal and the roof shale; there are also small faults, slips, and rolls, and places where the coal has been eroded and the space has been filled with drift. It is difficult, therefore, to calculate the total tonnage and to project any plan of operation. In many places the coal adheres to the roof and separates from it with difficulty. In one mine about an inch of coal is left to protect the roof shale from the air. In most mines the floor consists of a dark gray clay which heaves badly when wet.

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* Andros, S. O., "Coal Mining Practice in District III," Ill. Coal Min. Invest., Bul. 9, pp. 11 et seq., 1915; "Coal Mining in Illinois," Ill. Coal Min. Invest., Bul. 13, 1915.

Operations are conducted on the unmodified room-and-pillar system or on the so-called panel system. Dimensions of workings are given in Table 3. There are also four mines in the district which are operated on the long-wall system. Mining methods have not been given very careful attention, and the variations in the coal bed tend to minimize the effect of such attention as has been given. The method of mining generally practiced in the district involves the running of parallel main entries from the shaft toward the boundaries, and the turning of cross entries from the main entries at intervals of 350 to 400 feet. Rooms are turned off these cross entries on 30-foot to 42-foot centers, and are driven 20 to 30 feet wide. Room pillars average 9 feet in width and rooms 26 feet, but pillars are gouged as the miner pleases. This haphazard method is productive of so many squeezes that in some mines a modification of the system has been employed in which stub or room entries are turned off the cross entries. This method approaches the panel system and is called, locally, " block-room-and-pillar." Sometimes a sufficiently large cross barrier pillar is left to confine a squeeze to the block in which it originates, but generally the barrier pillar is gouged and squeezes ride over it unchecked until they reach a horseback or some ungouged pillar which is large enough to stop them. In several mines squeezes originating in rooms have traveled to the main barrier pillar and to the solid coal at the entry face. In one mine an entry was saved from a threatened squeeze by very heavy timbering ahead of the squeeze.

Eleven of the sixteen mines examined are at present operated on this semi-panel system, but the relative dimensions of room and room pillar have not been changed from previous operations. These dimensions are not safe under the roof found in the district. Room width is not uniform, but rooms are narrowed to avoid horsebacks and widened again where the coal resumes its normal thickness. There is a temptation to get all the coal possible on the advance, because the numerous rolls make uncertain the total tonnage which can be extracted from any area, and the rolls interfere seriously with any projected plan since cutting through them is expensive.

Pillars are drawn in only a few mines, and in these drawing is not done systematically but is confined to shooting slabs off the thickest parts of the pillars. Room pillars are tapered to cross-cuts in nearly all mines. In one case an attempt was made to draw pillars, and a track was laid along the rib, but objections were raised by the miners to this position of the track, and the attempt was abandoned. Principally because of the insufficient pillar-width, the floor of fire clay heaves badly even when dry.*

Nineteen mines were examined in this district, and the estimates of the percentage of recovery furnished at seventeen ranged from 55 to 75 per cent, averaging 67.26 per cent. It is probable that most of the estimates are too high for, although the gouging of pillars tends toward high percentage of extraction, careless methods always result in the loss of much larger quantities of coal than is supposed. One company, which has given careful attention to the forms and dimensions of its workings, is extracting about 70 per cent of the coal. It is doubtful however, if the extraction throughout the district as a whole amounts to 60 per cent.

14. *District V.*—Bed No. 5 in Saline and Gallatin Counties lies at a depth of 25 to 450 feet, being nearest the surface along the southern portion of the district. The bed varies in thickness from 4 to 8 feet, and averages 5⅓ feet in Saline County and 4 feet in Gallatin County.

The roof of the No. 5 coal in this district is of shale which is sometimes laminated and interbedded locally with bone and stringers of coal for a distance of 3 feet above the seam. The roof usually contains many concretions of iron pyrites called "niggerheads." It breaks quickly when wide spans are left supported, and it is drawn when it shows a plainly marked parting not more than 4 inches above the coal; but such a parting rarely occurs and the coal bed is so thin that the top coal cannot profitably be left in place. There are numerous falls which can be avoided only by making entries narrower than at present.

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* Andros, S. O., "Coal Mining Practice in District IV," Ill. Coal Min. Invest., Bul. 12, pp. 15 and 19, 1915.

The floor is of fire clay which in places contains much sand and heaves badly when wet. The bed contains many hills and rolls causing grades as high as 15 per cent in the entries of some mines. The coal is not pinched out at these hills, but follows the contours with undiminished thickness. In some mines about 9 inches of bottom coal is left below a "blue band," but as this bottom coal is not of good quality, increased facility in shooting compensates for the loss of coal.

The room-and-pillar system of mining is used exclusively, a main haulage entry and a parallel air-course being noted in every mine examined except one,—in which triple main entries were driven, two for intake air and one for return air and haulage. In the smaller mines and in many of the larger ones, the dimensions of workings are not suited to the roof conditions. The main entries vary in width from 14 to 16 feet. A few shaft pillars have been gouged. The room stumps, which are left when rooms are turned off the cross entries, are generally small. The closing of entries by roof falls may often be attributed to local squeezes which ride over the room stumps. Table 3 gives dimensions of workings for each mine examined.

The custom of driving wide rooms and entries, of leaving narrow pillars throughout the mine, and of obtaining all the coal possible on the advance without attempting to draw pillars has resulted in a high percentage of extraction for Illinois mines. The percentages given in Table 3 were calculated from the most nearly exact data obtainable at the time of their publication but are unquestionably too high. This reported extraction, averaging 67.1 per cent for the seven mines examined, was accomplished only with greatly increased expense for cleaning up.* One of the large operators of this district reports an average recovery at ten mines of 60.5 per cent over a 5-year period with a maximum of 72 per cent and a minimum of 52 per cent where the cover varies from 60 to 414 feet. Pillar drawing is not practiced, and it would be impossible to gain the percentages of coal given in the table if the dimensions given were adhered to, but pillars are gouged to such an extent that there should be a higher percentage of extraction than is calculated from the dimensions of rooms and pillars in the table.

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*Andros, S. O., "Coal Mining Practice in District V," Ill. Coal Min. Invest., Bul. 6, pp. 9 and 12, 1914.

15. *District VI.*—This district has experienced a rapid development, because the No. 6 coal commands a ready market; consequently mining on a large scale is possible. Bed No. 6 lies close to the surface along the Duquoin anticline* but dips sharply to the east, reaching a depth of 726 feet at Sesser. A general uplift has brought it to the surface along an east-west line extending through Carterville to Marion and along a southeast line from Marion to the boundary of the district. East of the area affected by the Duquoin anticline, the bed has a pronounced dip to the north. Along the outcrop line there are a few slopes and strippings, but the steep dip of the bed leaves only a small acreage with thin cover, and the remaining openings are shafts. The seam itself is thick, ranging from 7½ to 14 feet and averaging, as shown by 130 borings, 9 feet, 5 inches. A clean persistent parting of mother coal lies 14 to 24 inches below the top of the bed, and a second parting generally appears 5 to 8 inches lower down. Above the upper parting the coal occurs in layers 3 to 6 inches thick, with partings of mother coal between them.

The immediate roof consists of a gray shale 15 to 110 feet thick. This shale does not stand well when the coal is removed, and the top coal is generally left as a roof, at least until the rooms are finished. The bottom is generally of clay, four inches to eight feet thick, below which is limestone. There is only one persistent band of impurity in the bed. This, which is known as the blue band, generally consists of bone or shaly coal and is found uniformly at a height of 18 to 30 inches from the bottom. Its thickness varies from ½-inch to 2½ inches.†

The large number of squeezes which have occurred in mines of District VI would seem to indicate the presence of one or more thick beds of strong rock among the overlying strata. A study of the logs of numerous wells does not, however, show the presence of any continuous strong bed which would be a serious obstacle to the introduction of methods allowing a larger percentage of extraction. The State Geological Survey makes the following statement concerning the overlying limestone: "Over a large part of the area within 25 feet of the coal is a limestone cap rock which in places rests upon the coal, except for the draw slate that lies between. Where the limestone cap rock is not present within 25 feet of the coal it may be entirely absent, or lie at a considerably greater distance above the coal, amounting in some places possibly to as much as 100 feet."* The limestone cap rock is of variable thickness up to about 11 feet, the average thickness being 4 to 5 feet.† In some places sandstones are found at various distances above the coal, but none of these seems to be close enough to the coal to affect the choice of a mining method. In other words, it seems that there is no layer of rock, sufficiently near the coal to require serious consideration, which cannot be broken by careful attention to the proper methods. An examination of bore hole records of the Connellsville district of Pennsylvania, where the percentage of coal extracted is very high, indicates that there is more difficulty in breaking the overlying layers of rock in that district than would be experienced in most cases in District VI of Illinois. At a few mines an unusually wide room pillar is left in the middle of a panel for the purpose of limiting the area affected by a squeeze.

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*Andros, S. O., "Coal Mining Practice in District VI," Ill. Coal Min. Invest., Bul. 8, p. 11, 1914.

†Shaw, E. W., and Savage, T. E., U. S. Geol. Sur., Folio No. 185.

According to Table 3, all mines in the district, except strippings, are worked by the room-and-pillar method or by the panel method. Where the latter is employed, frequently no attention is paid to panel pillars so that the advantage of this method in the stopping of squeezes is largely lost. Practice is not uniform in regard to the number of rooms, which may be as low as 14 or as high as 30, turned from a room entry. The description of mining practice in this district‡ given in Bulletin 8 of the Coöperative Investigations says, "The immediate roof overlying the coal falls in slabs after short exposure to the air and top coal is usually left to protect it, but the cap rock is a tough coherent shale which does not break easily. The first mines opened in the district had widths of rooms and pillars unsuitable for this tough cap rock. New mines as they were opened adopted the dimensions of the older mines and a great waste has resulted through the loss of pillar coal. It will never be possible in this district to draw any considerable portion of the pillars where rooms 20 to 29 feet wide are driven with narrow room pillars. Fear of yardage charges has been an important factor in maintaining the present improper dimensions. . . . With present dimensions when rooms have been driven 200 to 300 feet there is a large area of unsupported cap rock. If an attempt is made to draw pillars under such conditions a squeeze is usually started which often rides over room and entry pillars and sometimes affects a large acreage. In one mine 85 acres were squeezed; in another, 80."

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*Cady, Gilbert H., "Coal Resources of District VI," Ill. Coal Min. Invest., Coöperative Agreement, Bul. 15, p. 83, 1916.

†Ibid, p. 82.

‡Andros, S. O., "Coal Mining Practice in District VI," Ill. Coal Min. Invest., Bul. 8, p. 12, 1914.

Early operations were carried on without regard to the possible production of squeezes. Pillars were gouged out or entirely removed whenever the demand for coal seemed to excuse this procedure; a natural consequence was the occurrence of squeezes. At one mine there have been five squeezes of which two involved about 80 acres each, one about 40, one about 20, and one possibly 10. The present plan for the future operation of this mine contemplates leaving barrier pillars 150 feet wide along the important entries, and removing this pillar coal later. It is believed that this plan will confine roof movement to the worked out areas and that the entry pillars and barriers can be extracted later. This plan is much the same as that shown by Fig. 32, page 102.

At another mine a large squeeze approached within 125 feet of the air shaft and caused a depression on the surface which necessitated the regrading of a considerable amount of track, including the track scales. Practice at this mine represents one extreme, since no attempt is made at room-pillar drawing beyond driving cross-cuts about 30 feet wide at the ends of rooms, and as much coal as possible is taken on the advance. An attempt will be made to take out barrier and entry pillars on the retreat. Rooms are driven 25 feet wide on 45-foot centers, and cross-cuts are 25 feet wide. The excavated area is about 50 per cent and the top coal, which is only 18 inches thick, is left up.

The figures for recovery of coal given in Table 3 are unquestionably too high although they were based on the best information available at the time. The average percentage of extraction in District VI is not more than 50 per cent, and it is probably nearer 45 per cent. The maps of mines may show an excavated area of 50 per cent or even more, but they do not take into account the unmined top coal. The thickness of coal taken out is generally about 7 feet and top coal ranging from a few inches to 4 or 5 feet in thickness is left. Even if the top coal is ignored, the extraction is not so high as the estimates generally indicate because of losses in squeezed areas and boundary barriers.

Special investigations on the subject of recovery made at several mines in Franklin County gave results which are summarized as follows:

> At one mine, the recovery in worked out areas where pillars are not drawn is about 65 per cent; where the pillars are taken, it is about 75 per cent.
>
> At another mine, close observations were made in connection with a study of subsidence. In a panel where the extraction was considered good and possibly above the average for the mine 40 per cent of the coal is left as pillars. Two feet or 20 per cent of the thickness of the bed is also left as top coal, and the loss from this cause would be 20 per cent of the remaining 60 per cent or 12 per cent of the total. The total loss is then at least 52 per cent. No attempt has been made to extract room pillars, but some entry pillars are taken, and top coal is taken over the area in which these pillars are drawn.
>
> At one of the mines where the thickness of the coal is greater than the average, little pillar work has been done. The coal varies from 9½ to nearly 16 feet in thickness,* and about 9 feet of it is taken out. Generally about one foot of coal is left on the bottom to avoid the possibility of taking up a bed of "black jack" which is not easily distinguishable from coal. This black jack is probably a coal of very high ash content. The leaving of bottom coal results in the elevation of the working place in the bed so that the top coal left is only 3 feet or even less in thickness. In a few places in this mine some pillar coal has been taken out. Where this was done, break-throughs about 24 feet wide were driven at the ends of the rooms. Then work was commenced at the ends of the pillars, and coal was taken out by pick work. This work seems to have been successful, but it has not been followed systematically. The largest number of pillars which have been taken together was six, and no attempt was made to obtain a break in the roof.

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*Cady, G. H., "Coal Resources of District VI," Ill. Coal Min. Invest., Bul. 15, p. 58, 1916.

The leaving of coal on the bottom is a practice followed at only a few other mines. In some cases where the blue band is thick the mining is done above it, and a portion of the upper part of the bed, ordinarily included in the top coal, is taken down. At two mines where this method is followed in part, the blue band and the coal below it are left in where the blue band is thick and the top coal is taken to within about ten inches of the roof, at which point there is a parting. Greater care is required to prevent the breaking of the top coal where this is done.

At one of the mines in the southern part of Franklin County a little pillar coal is drawn, though pillar coal is not depended upon for an important part of the output. Rooms are 25 feet wide with 20-foot pillars. Rooms are holed through into those of adjoining panels. When the rooms have reached their full lengths, cross-cuts 24 feet wide are driven across the ends of the pillars. In addition to these cross-cuts the pillars are probably slabbed to some extent. The coal is about 9 feet thick, and about 1½ feet of top coal is left up. No bottom coal is left. The barrier pillars are about 100 feet thick. Break-throughs are 21 feet wide. It seems hardly proper to speak of this kind of work as the extraction of pillar coal, but it represents a practice which is common in this district.

At one of the mines, pillars are drawn, beginning in the middle of a panel, in six rooms at a time; then another group of six pillars is attacked, one pillar being left untouched between the groups. This is simply another method of attempting to get as much coal as possible before being driven out by a fall of the top, and is not an attempt to break the cap rock.

Systematic work in the recovery of pillar coal as done at one of the mines is illustrated in Fig. 4. The mine is operated on the block system commonly called the panel system, though the panels are not kept sufficiently isolated to warrant the use of the term. Cross entries are driven at intervals of 1,370 feet, and panel entries are driven through from cross entry to cross entry. On each pair of panel entries twenty-eight rooms are turned to each side on 40-foot centers. A barrier pillar 125 feet wide is left along the cross entry. After the room entry is driven the rooms are necked, but only the first fourteen rooms on each side of the room entry are worked, and these are finished before the rooms at the other end of the panel are driven. Break-throughs are normally 11 feet wide; those at the ends of the rooms are 24 feet wide. Pillar drawing is commenced when the rooms at one end of the panel are finished, and the coal is taken out through the

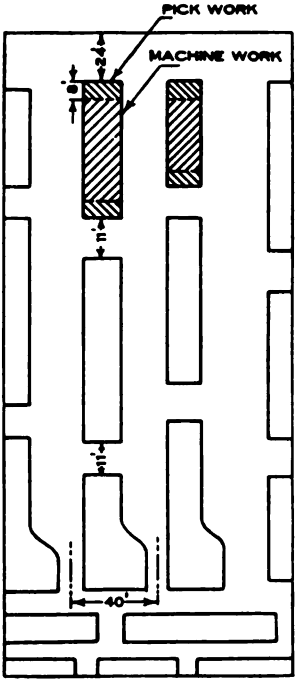

FIG. 4. PILLAR DRAWING IN FRANKLIN COUNTY, ILLINOIS

first cross entry, that is, the one next to room No. 1. The coal from the remaining portion of the panel is taken out through the next cross entry, that is, the one next to room No. 28. The advantage of this method lies in the fact that the extraction of the coal from the second half of the panel is not interfered with, as far as haulage and ventilation are concerned, by movements caused by pillar drawing in the first half. The pillar coal is attacked first by the driving of 24-foot cross-cuts at the ends of the rooms; then other cuts are made in the pillar

> with the breast machine in such manner as to leave stumps between the cuts and the break-throughs. These stumps are removed as far as possible by pick work, but the miners are not always able to finish as much work with the machines as is desired since this work is frequently interrupted by movements of the roof. As much of the coal as is possible is then taken out with picks. While the work at this mine is as systematic as that at any mine in southern Illinois, the company has no exact record of the amount of pillar coal extracted, but it is known that the pick-mined coal amounts to approximately 10 per cent of the output. When the pillars are not drawn, the recovery is estimated by the operators to be about 65 per cent; when they are taken, the estimates run about 75 per cent.

Through the courtesy of the Franklin County Coal Operators' Association (Illinois), data have been made available regarding the extraction of coal in that county as presented in the following paragraphs:

> The coal mined is the No. 6 bed of the State Geological Survey classification. Measurement of 113 sections taken in twelve of the largest mines in the county gave an average thickness of 9.2 feet of coal, the average minimum thickness for the same twelve mines being 8 feet, and the average maximum thickness 10.64 feet. The blue band, which is characteristic of the No. 6 coal bed, varied from ¼ to 2 inches in thickness, and its average distance from the floor was 21.5 inches. Owing to the difficulty of keeping up the shaly material above the coal bed, the top coal is almost universally left as roof protection, and, up to the present time, very little of this top coal has been recovered, although some operators are expecting to recover it at a later date in connection with pillar drawing. In one of the twelve mines from which the data were obtained, top coal was not left in the rooms. This, however, is exceptional practice, the average thickness of the top coal left in the twelve mines being 1½ feet. The average thickness of coal mined was 7.46 feet, and the average tonnage per acre to January 1, 1916 was 6,627 tons. This is equivalent to 40.7 per cent extraction, if it is assumed that all the 9.2-foot bed is available for ship-

> ment, or to 41.6 per cent if it is assumed that the blue band and refuse discarded in the loading, or 0.2 foot, is deducted from the thickness of the bed. A very careful estimate for each of the twelve mines noted, made by dividing the total amount of coal in the area mined up to January 1, 1916 into the actual shipments since the mine began operating, gave percentages of extraction varying from 37.7 to 49.5, or an average of 41.4 per cent.*

For six of the twelve mines, data were available for the average percentage of extraction in the portion of the bed actually mined; that is, the total thickness less the top coal left up to protect the roof. This average is 48.65. These mines are all comparatively new mines, and in only a few cases has any portion of the workings reached the boundary so as to permit drawing the pillars in return workings. At many of the mines it is hoped to increase the percentages of extraction through subsequent pillar drawings, but the amount of such increase is, of course, problematical. In many instances squeezes have already occurred, but as a general thing only the room pillars have been affected.

The twelve mines under discussion are representative of the practice in Franklin County and to a great extent of that of southern Illinois. In a number of these mines experiments are now being conducted to determine in what respect present methods of working may be modified to yield a larger percentage of extraction. Although these mines are operating under practically the same physical conditions and all on the panel system, the variation in the detailed operations, such as the number of rooms per panel, or the width of barrier pillars, indicate the necessity for a critical comparative study of details to determine the best method for the given conditions.

Investigations in Williamson County supplied the following facts:

> At one mine rooms are 21 feet wide on 40-foot centers. From 2 to 2½ feet of top coal is generally left up on the advance. In one part of the mine the coal is 11 feet thick and only 7½ feet of it is taken out. It is estimated that 40 to 50 per cent of the pillar coal is won. The top coal, which is the

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* The Peabody Coal Company reports that the percentages of extraction at its four mines in this district are 67, 63, 55, and 55.

best part of the bed, is taken out when the pillars are drawn. There are no definite records on recovery, but it is probable that 55 or 60 per cent is gained. If 7½ of the 11 feet are removed, and 40 per cent of the pillars and all the top coal over the area in which the pillars are drawn are removed, the extraction is about 60 per cent.

In another mine a rather high percentage of extraction is attained because of favorable conditions which permit the leaving of small pillars. The coal is 9 feet thick and the depth only 100 feet. Rooms are 24 feet wide on 35-foot centers. In some cases top coal, about 20 inches thick, is left if the machine runners think the top is insecure. Probably from 65 to 70 per cent of the coal is taken out. No pillar work could be done with rooms and pillars of these dimensions, but on one side of the mine 15-foot pillars are now being left with the intention of taking them out on the retreat.

At some mines in the western part of Williamson County considerable trouble has been experienced, because large quantities of water enter when the top is broken. The cover here is only about 100 feet thick and there are only 3 to 4 feet of solid rock. The rooms of one mine are 20 feet wide and are driven on 40-foot centers, although they are sometimes crowded. Some rooms were driven on 32-foot centers, but the pillars were not sufficient to prevent squeezes. Entries are 12 feet wide and entry pillars 20 and 25 feet wide. The coal is 5 to 11 feet thick with an average thickness of 8 feet. Top coal averages 20 inches in thickness. Above the bed is a shale; above this is a so-called soapstone, ranging from 2 to 8 feet in thickness and averaging about 4 feet; and above this is a black shale. In some places a draw slate from 1 to 2 feet thick occurs above the coal, and above this is limestone 1 to 3½ feet thick. Where is no draw slate, there is no limestone. An unconsolidated sand is found in some places above the coal. The pillars are sometimes slabbed a little to compensate for the coal left in entry and barrier pillars. When this slabbing is done, the extraction amounts to about 50 per cent. In some cases extra cross-cuts are taken, and the extraction is thereby increased to about 75 per cent. On the whole, the extraction is estimated to be about 60 per cent,— an estimate which is probably reliable since the work is more carefully done here than at many other mines of the district. Where more than 60 per cent of the coal is taken out in this mine, the roof breaks and water enters in large quantities. Although the amount of water is influenced by precipitation, the flow is continuous. In one case where the rock had been broken and a large quantity of water had entered the mine, it was thought that the strata was drained to some extent and that it would be possible to allow the top to break at a slightly higher elevation. It was found, however, that the new break allowed a large amount of water to enter the mine, and it has been impossible for the company to do any pillar work. It is planned, as some of the workings reach the boundaries, to draw pillar coal. In these cases pumps will already have been installed, and it will be possible to conduct the water to these. The water, moreover, will be entering in abandoned places and not between the workers and the shafts. Some of the workings are now not far from the boundaries, and the plan can be put into operation in the near future.

At another mine the coal is 9 feet thick, the top is of white shale, and the bottom of fire clay. The top coal is about 2 feet thick. The mine is operated on the panel system. No pillars are drawn until the rooms on an entry have been finished; then a cross-cut three machine-cuts wide, or about 20 feet, is driven at the ends of the pillars in about half the rooms on the inside end of the stub. Following this, another cross-cut is made farther back in the pillar leaving a stump about 10 feet wide. The distance between the first and second cuts varies according to conditions, or according to the judgment or inclination of the machine men. If it is made farther back, some of the pillar coal is lost. This operation is repeated until the first break-through in each pillar is reached, the remainder of the pillar being left standing until all the rooms on the stub are finished; then the room stump and the entry pillars are drawn. No effort is made to obtain a break in the roof, and the leaving of stumps of pillars is likely, by partially sustaining the roof, to bring on a squeeze. At present the driving of rooms without necks is being tried, the purpose being to avoid payment for narrow work; and it is believed there will not be sufficient difference between the support left under that system and that left under the present system to endanger the entries. These rooms without necks are turned six machine-cuts wide and are widened to seven cuts beyond the first cross-cut.

At another mine the average thickness of the coal is 9 feet, 4 inches. The top coal is about 2 feet thick and is left up until the pillars have been partly drawn back, being taken down just before the track is removed. There is generally a good parting between the main bed and the top coal, which is said to be poorer than the main bed. Above the bed is shale of unknown thickness, which has never broken high enough to expose any other rock above it. This shale slacks when exposed to the air. It does not form a very good top and most of the entries are timbered. The bottom is generally of clay, but in some places limestone appears next to the coal. Rooms are 20 feet wide on 30- to 35-foot centers and are 185 to 190 feet long. Stub entries are turned on 400-foot centers, and 16 to 18 rooms are turned from a stub. The room pillars are gouged to a considerable extent. It is planned that all the rooms in a panel shall be driven to their full length before pillar drawing is commenced, but this plan is not always followed, and squeezing sometimes commences before all the pillars can be attacked. This, of course, is promoted by the gouging of room pillars. When the rooms are finished, cross-cuts 20 feet wide are driven at the face with breast machines. The rest of the pillar work is generally done with picks though machines are used when possible. Movement of the roof, however, generally interferes with machine work after the first cross-cut. The pick work generally consists of slabbing along the sides of pillars, but machines are sometimes used. Squeezes have always been confined to the panels, and no entries have been lost until the entry pillars have been drawn. A careful computation, based upon a comparison between the actual area worked and the number of tons hoisted, shows that the extraction at this mine has been 48.89 per cent. This is one of the most thoroughly worked mines in the No. 6 bed, and the estimation of percentage of extraction is undoubtedly as close as any that has been made. The results found furnish one of the reasons for the statement that extraction in most mines of the district is less than the operators of the mines believe it to be.

At a Perry County mine the general system is the same as in Williamson County. Room entries are driven through from cross entry to cross entry. A somewhat closer adherence to the panel system is to be noted, however, in that 25-foot pillars are left at the ends of rooms. Rooms are 24 feet wide on 60-foot centers, and they are driven 250 feet long. Break-throughs are staggered. When the room is completed, an 18-foot cross-cut is driven through the pillar at the end. Top coal, about 3 feet thick (the best of the bed in quality* as it is at the mine last mentioned), is left up until pillar drawing commences. Pillar drawing is commenced in the middle of the panel. After the completion of the cross-cut at the end of the first pillar attacked, the top coal is loosened by a light shot near each rib. Work on the pillars is then prosecuted by making a cut through the pillar, if the condition of the top will permit, wide enough to leave an 8-foot stump at the end of the pillar and another of the same dimension next to the nearest break-through. These two 8-foot stumps and whatever is left by the machines are taken out by hand work. Two men are used on solid work and two on the machine. This method has been found successful and a considerable amount of pillar coal has been recovered, but pillar drawing is not a necessary part of the system and is not always carried out. Where the pillar coal and the top coal are taken, the recovery is said to be from 75 to 80 per cent of the coal in the area actually worked.

Several plans are now being tried or considered for the more nearly complete extraction of the coal in this district. One of these, which has so far been given only an incomplete test, is a panel long-wall method. Double entries, which would have been the room entries of a panel under the ordinary methods of operation, were driven 340 feet long. At the end, two rooms were driven on each side separated by 25-foot pillars. The rooms at the extreme end of the block on each side were 9 feet wide and the ones further back were 18 feet wide; each was 200 feet long. They were connected at the ends so that ventilation was obtained, the course of the air current being as shown in Fig. 5. Then the outby ribs of the 18-foot rooms were worked as long-wall faces by continuous-cutting chain machines making a 6-foot cut. The top behind the working face was propped. It was the intention to support the immediate roof until the face had advanced some distance and then to make an attempt to break the overlying rock by the withdrawal of the props. This plan was found to be impossible, however, as the top fell when the face had advanced only about 40 feet. Other conditions made it necessary to discontinue the experiment temporarily. In operating by this method, sprags were placed in the cutting behind the machine to prevent the premature fall of the coal. No trouble was experienced in getting the coal down; it was produced very rapidly and was easily handled. At present it is not known whether the top can be broken along the desired line, but it will be seen that this line is only 400 feet long and that it is interrupted in the

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*For a discussion of the differences between the top coal and the remainder of the bed see "Chemical Study of Illinois Coals," by S. W. Parr, Ill. Coal Min. Invest., Bul. 3, p. 49, 1916.

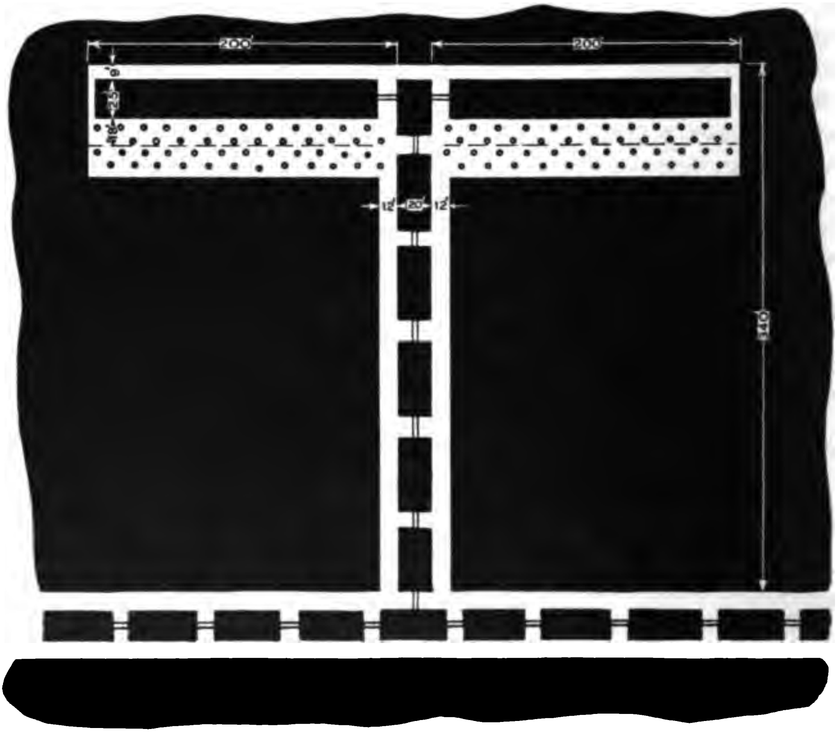

Fig. 5. Panel Long-wall

center by the entry pillar. Even if it is not possible to break the top and to work the coal back continuously on two longwall faces, it seems that the attack can be repeated farther back in the block and that coal can be produced as cheaply as by the ordinary method; also that a much higher percentage of extraction can be attained. If it should be necessary to follow the method by repeated attacks on the block, there would be some resemblance to the "single room" method successfully worked in West Virginia.

Various other plans for higher extraction have been suggested and some have been partly applied, but the great demand for coal, which has been stimulated by the European War, has caused coal producers to concentrate all their attention upon the immediate production of a large tonnage. Anything in the nature of experimental work will be postponed until the return of more nearly normal conditions, but there is reason to believe that successful efforts will be made to increase the percentage of extraction and that the present large loss will be greatly decreased.

16. *District VII.*—The coal worked in District VII is the No. 6 bed on the west side of the Duquoin anticline. The thickness varies from 2½ feet to 14 feet and averages about 7 feet. There is a well defined parting plane in the coal about 18 inches from the roof. Where the roof is of black shale and where the coal is 7 feet or more in thickness, the upper bench or "top coal" is left. The roof is a non-calcareous black shale, a calcareous gray shale called locally "white top" or "soapstone," an unconsolidated dark gray or black shale called "clod" and made up of fragments of varying size and hardness extremely difficult to support, or a hard gray limestone called "rock top." A poorly defined cleat or cleavage in the coal may be seen in some places. The floor throughout the district is of fire clay which generally heaves when wet.

The thickness of the coal is almost ideal for easy working and for large production; some of the mines have obtained daily capacities which rank among the highest in the world. The older mines have been worked without much regard to system, but the newer ones are more carefully planned. The planning, however, is directed toward large daily production rather than toward a high percentage of extraction.

Varying roof conditions often make different entry and room widths necessary in different sections of a mine. In many mines the entries and rooms under rock top are too wide and the pillars too narrow,—a condition responsible for squeezes which sometimes have endangered even the shaft. Squeezes have occurred in thirteen of the twenty-five mines examined in this district; they have generally begun in sections in which the roof was of limestone. In mines in which the rooms are not frequently surveyed there is no definite knowledge of room-pillar width except at cross-cuts. Table 3 gives dimensions of workings at each of the mines examined.

In ten of the mines examined where the immediate roof was of thick black shale, top coal was left to prevent variations of temperature and humidity from affecting the shale of the roof proper, which spalls badly when exposed to the air. Where no top coal is left, this black shale usually falls with the coal or is drawn. Where there is less than four inches of shale between the coal and the limestone, the shale is drawn. In some mines where the latter is more than four inches thick it is propped; in others it is drawn, unless it is more than two feet in thickness.* The Peabody Coal Company reports extractions at its mines in this district of 65, 62, 60 and 50 per cent. At the twenty-five mines examined, the average estimated percentage of recovery furnished by the operators was 55.5 per cent. So far as is known, no efforts have been made to extract a higher percentage of coal. The reason for this attitude is to be found partly in the condition of the surface, which in many places is so nearly level that any noticeable subsidence disturbs the drainage and effects the value of the surface for agricultural purposes.

At present the abandonment of a large percentage of the coal is a result of the difficulties experienced in attempting to secure satisfactory agreements with the owners of the surface. In some cases the owners of coal rights are not the owners of the surface and are not free from responsibility for surface damage. Since the operators have found that the estimates of damage caused by subsidence are likely to be very high, it has become the custom to operate the mines under methods which will avoid subsidence. Unfortunately it has not been possible to estimate the exact amount of coal which must be left in the ground, and squeezes and subsidences have sometimes occurred when it was thought that sufficient coal had been left.

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*Andros, S. O., "Coal Mining Practice in District VII." Ill. Coal. Min. Invest., Bul. 4, p. 11, 1914.

Even where low value of the land or good drainage reduces the cost of possible injury to the surface by subsidence, no effort is made to secure a higher extraction. The occurrence of squeezes is feared, and experience shows that the only way to prevent them without radically changing the system of mining is to leave large amounts of coal in the form of pillars. One company which was formerly getting 50 to 60 per cent of the coal with frequent squeezes and subsidence has changed the dimensions of rooms and pillars so that now only 40 to 50 per cent is obtained. Thus far, with the new dimensions, squeezes have not occurred. No effort has been made to extract pillars systematically with the purpose of breaking the cap rock and thus preventing a squeeze by relieving the stress on the pillars, but there is nothing to indicate that this plan could not be carried out.

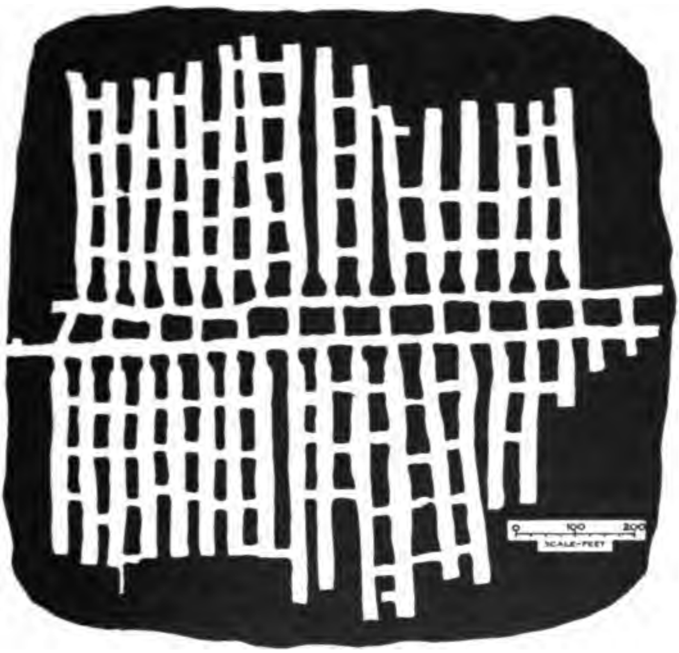

FIG. 6. PLAN OF AN OPERATION IN MACOUPIN COUNTY, ILLINOIS, SHOWING EXTRACTION IN A LIMITED AREA

The plan of a portion of one of the mines is shown as Fig. 6. This may be taken as a fairly typical projection of large mines in this district. In many of the mines, including parts of the one illustrated, the workings are on a panel system, but probably not enough attention is paid to the matter of leaving pillars sufficiently wide to prevent the spread of squeezes beyond the boundaries of the panels. This illustration is presented, because the mine was surveyed with unusual care in connection with an investigation of subsidence which is being carried on by the Coöperative Coal Mining Investigations. The rooms and pillars are about 30 feet wide; this dimension was adopted with the belief that the top would be held up by pillars of this width left between 30-foot rooms. It had been found that the roof would fall if 25-foot pillars were left between 25-foot rooms. In the restricted area measured, the extraction amounts to 59.2 per cent of the area worked; that is, 40.8 per cent of the area has been left as pillars.

17. *District VIII.*—In District VIII, seams 6 and 7 are mined. In both seams there are numerous rolls of roof and floor called "faults," or "horsebacks." In many cases the roll completely displaces the coal.

Seam 6 averages 6 feet in thickness. Near Danville the immediate roof is of grayish black shale about 6 feet thick. This shale, lying between the coal and a cap rock of dark gray nodular limestone, makes a roof which is easy to support. In the vicinity of Westville and Georgetown, the immediate roof is generally of gray shale which shows no distinct bedding, has little cohesion, falls in conchoidal masses, and is extremely difficult to support. Stringers of coal, furthermore, extend from the seam proper into the roof material and render the task of supporting the roof more difficult. Occasionally there are 3 to 4 inches of black shale between the coal and the gray shale which forms the cap rock. Wherever this black shale is broken, air and moisture disintegrate the gray shale cap rock, and the roof becomes unsupportable. In all parts of the Danville district the floor is of soft fire clay.

Seam 7 varies in thickness from 2½ to 5½ feet, the average being 5 feet. The coal has two benches separated by a clay band one inch thick, which persists throughout the bed from 6 to 8 inches above the floor. This bed also has numerous rolls.

While the stripping operations, which are important in this district, are conducted in the No. 7 bed, the largest underground operations are in the No. 6 bed. The mines are operated on the room-and-pillar method, or a modification of it, but the numerous rolls in the roof prevent close adherence to the system. The frequent occurrence of rolls has a marked effect upon the manner of driving rooms. In a roll area it is difficult to support the roof, and the expense of driving through the hard rock of the roll is great; consequently, when a roll is encountered in driving rooms it is customary to change the direction

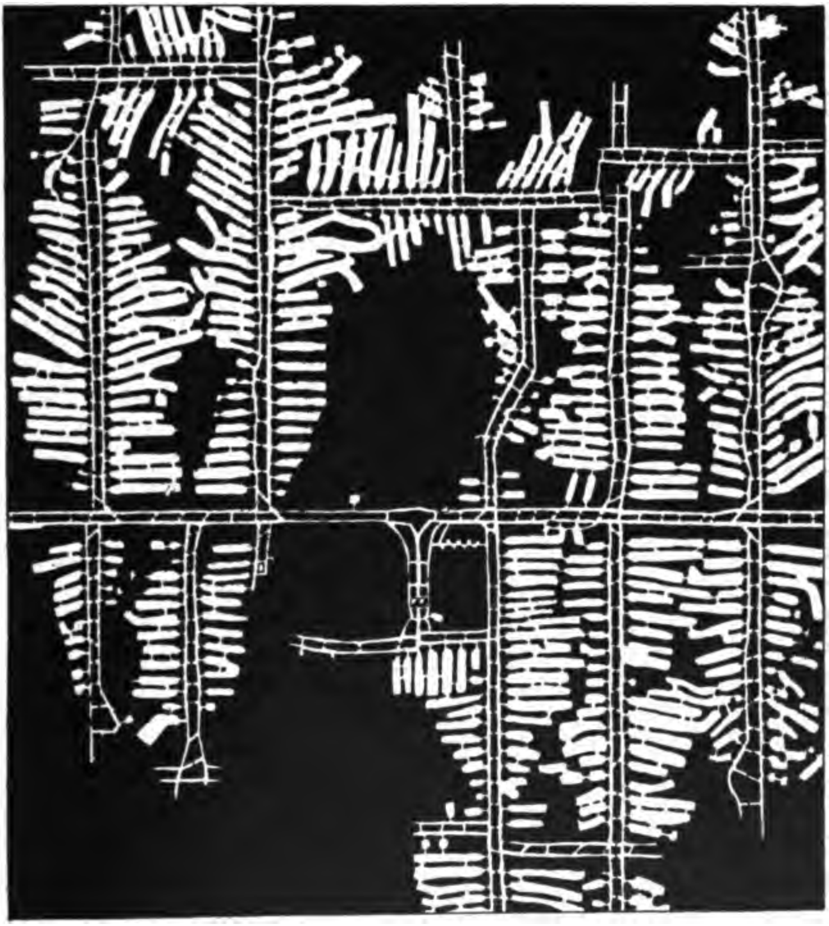

FIG. 7. PLAN OF MINE IN VERMILION COUNTY

of the room and to drive it parallel with the roll until the coal resumes its normal condition, as shown in Fig. 7, which is a map of a mine typical of the district. Often it is necessary to abandon a room before it has been driven to its proper length. Since the rolls are of frequent occurrence, the amount of coal that may be gained in any section of the mine is problematical; consequently, the operator, on reaching that portion of the coal where the seam regains its normal thickness, will attempt to get as much of the coal as possible during the first working. Little attempt is made to preserve a constant room-pillar width, and the practice of gouging pillars is common in the smaller mines.* No systematic pillar drawing is attempted, because with present practice there is little pillar coal left to draw when the rooms are driven to their full length. The roof is so treacherous, especially in the vicinity of the rolls, that it is not safe to leave wide spans of roof unsupported by pillars.

The width of room pillars at the mines examined varied from 4 to 16 feet, and room widths varied from 21 to 43 feet. Table 3 gives dimensions of workings at each mine examined. Very narrow room pillars were found in mine No. 91, where the following dimensions were recorded; room centers, 47 feet; room widths, 43 feet; room pillar width, 4 feet.

Although pillar gouging in the district has resulted in a high percentage of extraction from the bed in the first working, it has caused a subsequent loss of coal through squeezes due to narrow pillars. The average extraction for the six mines examined, as reported by the operators, is 70 per cent. Table 3 gives also the percentage of the bed extracted at each mine. These percentages were calculated from measurements made in the mines and were checked by records of production per acre obtained from the books of each operating company and by planimeter measurements of mine maps. The Peabody Coal Company reports an extraction of 66 per cent at its mine in this district.

At one of the mines, almost all the pillar coal was extracted after all the advance work had been done, and the roof was supported largely by the rolls which occurred at intervals of 60 to 100 feet. At another mine pillars are being extracted, and it is estimated, that the total recovery will amount to about 85 per cent.

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*Andros, S. O., "Coal Mining Practice in District VIII," Ill. Coal Min. Invest., Bul. 2, p. 16, 1914.

18. *Conclusion.*—It will be seen that nearly all the work in Illinois described as pillar drawing is unsystematic. It is merely incidental to the mining of room coal, and preparation for it is rarely made in laying out the mines. There are no apparent reasons, so far as physical conditions are concerned, except in a few instances, why plans could not be made for leaving pillars large enough to support the top during the advance work and for recovering the pillars on the retreat. Squeezes could thus be avoided, and the percentage of extraction could be increased materially. The commercial conditions which seem to make such a course difficult could probably be overcome, except in those cases in which subsidence of the surface subjects the operators to claims for damages in excess of amounts which would seem to be reasonable compensation for the injury done. The law covering payments for damages due to subsidence ought to be made so clear that there could be no doubt concerning the amount to be paid, and this amount should be limited to a fair compensation for the injury actually done.

At present there is promise of a considerable improvement with regard to the percentage of coal extracted from Illinois mines. The subject is receiving more and more attention on the part of coal producers and careful planning of the work with a view to high extraction as well as to low cost will follow as the natural result of greater interest on the part of the operators.

CHAPTER III

METHODS AND RECOVERY IN THE UNITED STATES

19. *Early Methods in the United States.*—This chapter presents a discussion of early methods of mining coal in the United States, information regarding the percentage of coal recovered in different districts, and descriptions of the most advanced methods employed for obtaining high extraction, especially those which are applicable to conditions in Illinois. In collecting this material all available sources of information have been utilized. The descriptions of methods have been taken largely from the technical literature of coal mining, but in instances in which the correctness of the description seemed in doubt, or in which statements concerning the percentage of extraction seemed to need verification, the subjects have been reviewed by persons familiar with the local conditions.

In response to the large number of inquiries sent out, many persons have furnished the desired information in as nearly complete form as possible, but in many cases there has been available no authentic information on the subject of recovery. The estimates are necessarily more or less approximate because the conditions are such that it is practically impossible to obtain correct values, or the subject has not been considered of sufficient importance by the operators to warrant the expenditure of the time and money necessary for obtaining the values. It is believed that the values for percentage of extraction given in the following pages represent the most reliable information obtainable on the subject, but they are not presented as being absolutely correct.

At the time mining was begun here, this country was a colonial possession of Great Britain; the methods of mining to which immigrants were accustomed were those of Great Britain, and the application of these methods to mining problems in America was a matter of course. The development of the early English methods is discussed in the appendix. The coal miners of this country, furthermore, have been for the most part men who received their training in the work in England, Scotland, or Wales, or children of such men, and not until a comparatively recent date did these miners loss their dominance in the American coal fields. The conditions under which coal was found in this country were also not very different from those in Great Britain. It was natural, therefore, that bituminous coal mining practice in this country should correspond to that of Great Britain at the time the industry began here.

Mining in this country was begun in the Richmond (Virginia) basin about the middle of the eighteenth century. There seems to be no clear record of the methods followed, but it is known that a pillar system was employed, and that, as the coal was reached in some places at a depth of several hundred feet, a considerable amount of the coal was left in the ground. It is said that the pillars were to be extracted on the retreat, but no definite record is found to indicate that this was done.

Western Pennsylvania was the next district to take up coal mining on an important scale. Maryland and West Virginia followed, basing their early methods for the most part on what had been done in Pennsylvania.

20. *Pennsylvania.*—The early history of coal mining in the western Pennsylvania district is typical of that in other sections of the country, having a similar hilly topography. When coal mining was commenced, an abundant supply of coal was found outcropping on the hills in the neighborhood of Pittsburgh, and these seams were attacked by numerous small mines on the outcrop. As the workings were extended under cover, the single entry system was followed, and as it was impossible to obtain good ventilation with this system, the rooms were driven to only a short distance, and the entry itself was not long. Later the double entry method was employed, in which two parallel entries were used, respectively, for intake and return air. Since the distance to which rooms could be driven was limited, it was impossible to work any large territory by this method; hence, as the size of the mines increased, the cross entry system was introduced. The underground developments were the same whether the coal was reached by drifts, by slopes, or by shafts.

Among the many experiments tried in the Pittsburgh bed was that involving the use of double rooms with double necks, or of double rooms with single necks, but the amount of timber required for posts made these methods too expensive and by 1906 they were in use in a very few mines. The long-wall system also was tried and abandoned.*

No record has been found of the time at which the drawing of pillars was commenced, and it is probable that this method was followed more or less from the beginning in such mines as were systematically developed.

Toward the end of the last century, the double entry system had been further developed by the turning of room or butt entries from the cross or face entries. In some mines a few of the entries were driven to the boundary, and then all the rooms were opened at once, but some of the center rooms would sometimes reach their limits before those whose pillars should have been drawn first. In other cases only the inby half of the rooms on each entry was turned first, while in still others one entry was completely exhausted before any side work was done on its parallel entry. In most cases, a room-and-pillar method was used with double entries, each about 9 feet wide. Main entries were separated by a pillar 51 feet wide with cut-throughs for ventilation. The main entries were driven on the butt of the coal, and face entries were turned from them about 1,000 feet apart. From these face entries, secondary butt entries or room entries were driven

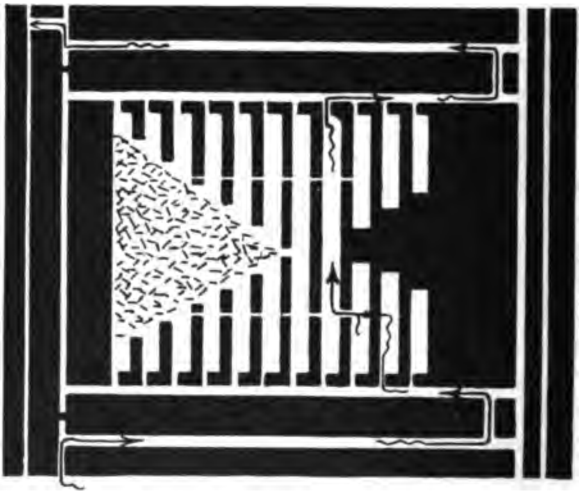

FIG. 8. OLD METHOD OF ROOM-AND-PILLAR IN PITTSBURGH, PA., DISTRICT

about 400 feet apart. Rooms about 20 feet wide and 200 feet long were turned on the face of the coal. The room necks were 21 feet long and 9 feet wide. Room pillars were 15 or 20 feet wide, according to the cover above the coal. The rooms were turned from the butt entries as fast as these were driven, room pillars being drawn as mining progressed.* The objectionable features of this method are:—poor ventilation, dangerous gob, entries filled with fallen dirt requiring expense for cleaning up, maximum extent of track for the minimum quantity of coal, thus greatly increasing the cost of animal haulage, loss of thousands of tons of coal, compulsory driving of narrow work in room turning, and squeezes, which damage the coal and greatly increase the hazard of mining.

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*Dixon, Charlton, "A New Method of Coal Mining," Mines and Minerals, Vol. 27, p. 32, 1906.

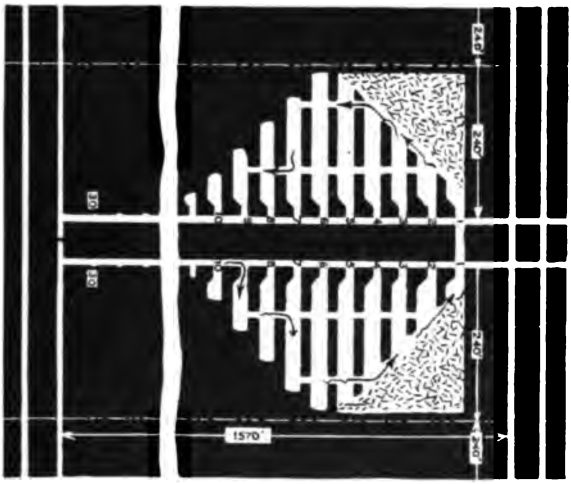

FIG. 9. IMPROVED METHOD OF ROOM-AND-PILLAR IN PITTSBURGH, PA., DISTRICT

The difficulty of ventilation becomes most serious when the rooms from one entry are holed through into those approaching from a neighboring entry (see Fig. 8). Often this will occur two-thirds of the distance up each of these entries; thus all the pillars below the short circuit are deprived of proper ventilation at a point where it is constantly needed. After the pillars are drawn and the roof falls, there is no appreciable movement of air through the gob, and it often fills with explosive gas.

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*Auchmuty, H. L., in Coal and Metal Miners' Pocket Book, 9th ed., p. 295.

A method described by Dixon, in the article previously referred to, was soon adopted with various modifications, although it is possible that it was already in use in one or more places at that time. According to this method the territory was laid off into blocks (see Fig. 9) 1,570 feet long, allowing for a barrier pillar 200 feet wide along the main entries and for another 200 feet along the next pair of face entries. A pair of room entries separated by a 54-foot pillar was driven through the center of this block, and thirty rooms were turned from each entry. Rooms were 240 feet long, about 26 feet wide, and were driven on 39-foot centers, thus leaving 13-foot pillars. Room turning was begun at the inby ends of the room entries, a reversal of the common practice of the time. The drawing of pillars was commenced as soon as the rooms were finished, and the line of break was kept at the proper angle by carefully timing the extraction of pillars. In this method the ventilation was considerably better than in the earlier method; but the air current, after passing through the district of pillar work on one room entry, went through the advancing rooms turned from the other. This difficulty was avoided in later methods by exhausting one room entry before room work was done on the other. The roof in the entries was easily maintained because the entries, with the exception of those on which rooms were being worked, were in solid coal. At the finishing of a block the minimum of track was in use for the minimum of coal passing over it. Track was not left in place awaiting the withdrawal of entry pillars; therefore it was not exposed to the corrosive action of mine water. Since the room pillars were attacked immediately, there was little danger of deterioration of coal or of trouble from falls. Most of the props could be recovered as they had not been subjected to any great pressure. Under the old system 50 per cent of the wood rails in rooms were lost while awaiting the attack on the ribs, and about 75 per cent of the posts were lost.

Referring to the conditions and the methods employed in mining, F. W. Cunningham* said in 1910:— "The operator in the Pittsburgh coal field, with the price of coal where it is to-day, must get the largest percentage of lump with the least amount of fine coal, and this by machine mining, in order that he may compete with coal operators in other fields." The rooms, therefore, are made as wide as possible to obtain the greatest percentage of lump coal, and the pillars are left as narrow as possible, because the greatest percentage of crushed coal comes from them. This fact explains why the use of narrow rooms and wide pillars, common in the Connellsville district, does not appeal to operators in the Pittsburgh district. It also explains the loss of much pillar coal, because a period of dull market results in the stopping of pillar work and only large coal from the rooms is marketed. A large number of rooms, accordingly, may be driven up to their limits without the extraction of room pillars, and the recovery of these pillars is unprofitable after the rooms have stood for a number of years.

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* Cunningham, F. W., "The Best Methods of Removing Coal Pillars," Proc. Coal Min. Inst. Amer., p. 275, 1910; p. 85, 1911.

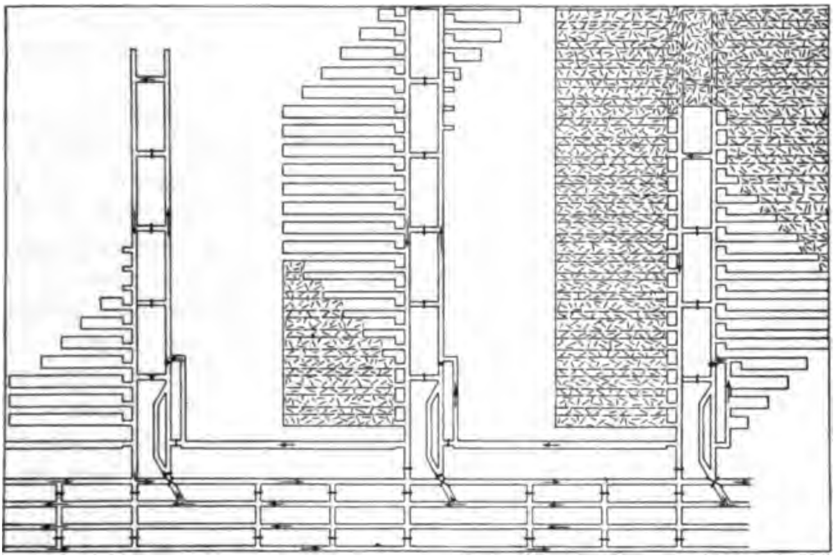

FIG. 10. MODERN METHOD IN PITTSBURGH DISTRICT

Fig. 10 illustrates the series of operations incident to one method of extraction of stump and chain pillars. In this method, rooms are turned and worked out progressively along one of a pair of room entries, probably the last, the pillars being drawn back as soon as the rooms are finished. There is thus a diagonal line of rooms advancing and another diagonal line, practically at right angles to this, retreating. On the other room entry of the pair, the driving of rooms is commenced at the inby end and proceeds outward. In some instances the entry pillars have been drawn on the retreat as illustrated in Fig. 10, and in others they have been left until all the rooms and room pillars have been finished. In the latter case it has sometimes been possible to obtain the coal from these entry pillars, but frequently all or part of it has been lost. The method illustrated in Fig. 10 was in use in the Pittsburgh district proper, that is in the high coal along the Monongahela River. Cunningham says that the extraction by this method would average about 80 per cent. Some companies, however, claim an extraction of 90 per cent. Some differences in percentages of extraction may be accounted for by the different methods followed in estimating: the whole bed from the limestone to the top of the seam may be taken into consideration, or the thickness of the slate partings may be subtracted.

One of the principal reasons for taking the rooms turned from one of a pair of butt entries on the advance and those turned from the parallel butt on the retreat was that this procedure made it possible to have the air current always blowing from the room work to the pillar work. This constantly moving current of air prevented gases set free by the pillar work from being carried to men working in advancing places. The miners in the pillar workings used locked safety lamps, while those in the room workings used open lamps.

Until about 1910, mining machines were used in the Pittsburgh district only in room and entry work, while pillar coal was undercut with picks. A method designed to permit the mining of pillars by machines is illustrated by Fig. 11. Cunningham gave the following facts concerning this method:—24-foot rooms are turned on 39-foot centers. After the room is worked out with a machine, a cut about 25 feet wide is made across the end of the pillar; then another cut of the same width is made far enough back on the pillar to leave a stump 5 to 8 feet wide, and the stump is removed by pick work after the machine work is finished. The stumps serve to protect the machine runners and the machine by supporting the top. It was said that at one mine where this system was used 70 per cent of the pillar coal was extracted with machines, and 30 per cent was pick mined. Good falls were obtained, and no ribs were lost. The recovery of timber was not so good as in the Connellsville region or in mines where there is no refuse gobbed along the roadways.*

An old and common method of working is illustrated by Fig.

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* For another description of the method in use about 1910 see Schellenberg, F. C., "Systematic Exploitation in the Pittsburgh Coal Seam," Trans. Amer. Inst. Min. Engrs., Vol. 41, p. 225, 1910.

12. Track is laid in the middle of the room, and the room pillars are made as narrow as possible after the room has advanced 100 feet, or to the first cut-through. Commonly no attempt is made to recover the pillar coal beyond this point, though it is often recovered nearer the entries by working the pillar along the side of the fall. One of the chief operators in the Pittsburgh district claimed a recovery of 90 per cent of marketable coal by this method, but it seems that such a recovery could be made only over an area of a few acres and that the recovery over the entire area of the mine would be much lower. Schellenberg expressed the opinion that the recovery would not be more than 60 per cent if an area of 10 acres were considered.

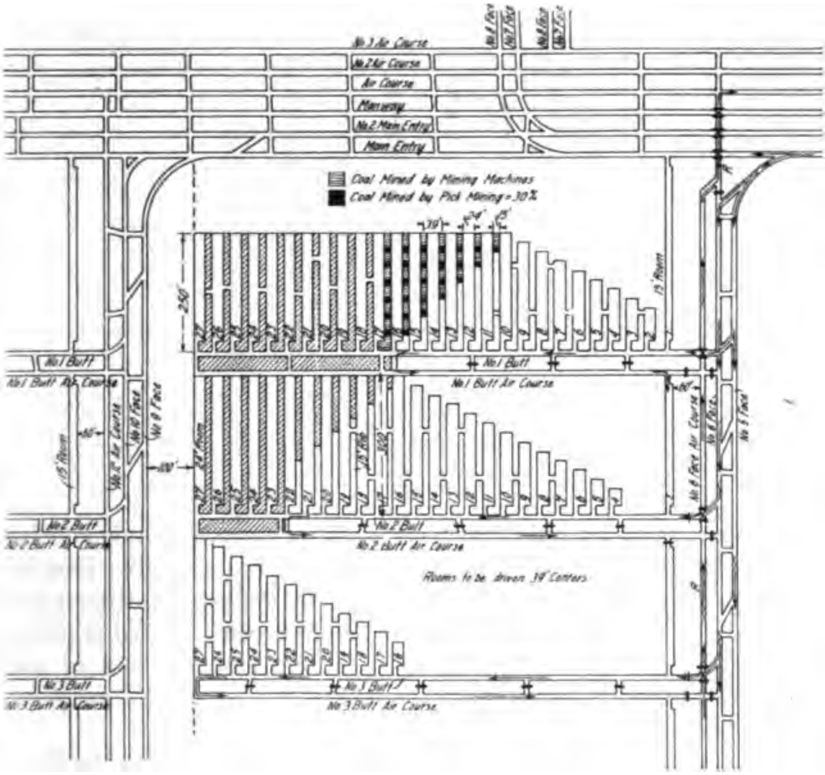

FIG. 11. PILLAR DRAWING WITH MACHINES IN PITTSBURGH, PA., DISTRICT

In the discussion of Cunningham's article, G. S. Baton said that entry pillars were not often recovered in the Pittsburgh district except under remarkably favorable conditions. In his opinion not more than 40 per cent of the entry pillars were recovered where there was much overburden. It is probable that a larger percentage than this is being recovered now in the more carefully operated mines. Another practice in pillar drawing which had been used in the Pittsburgh district and in other districts is illustrated by Fig. 13. A curtain of coal is left to keep out the gob when drawing room pillars, and the loss of coal amounts to about as much as it does where the pillars are narrowed and not drawn.*

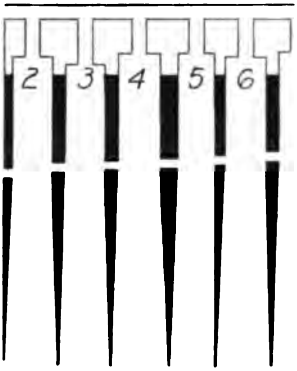

FIG. 12. TAPERED PILLARS

It has been seen that in most of the methods employed in the Pittsburgh district, the pillar coal is taken out by pick work, and it has been within only a very recent period that even the room coal has been undercut by machines. Machine mining of pillars is cheaper than pick work and operators have recently introduced this more advanced method wherever it seemed possible. The immediate reason for this action has been the increased cost of production, largely due to high wages and expenses caused by changes in the laws affecting mining, without a corresponding advance in selling price. Since the demand is still greater for the lump coal than for the smaller sizes, it has been necessary to increase the size of the pillars to prevent the objectionable crushing of the pillar coal. Another reason for increasing the width of the pillars is to be found in the practical difficulty of using machines on very narrow pillars. The thickness of room pillars varies, but the most common distances between the centers are 33, 36, 39, and 42 feet. With 33-foot room-centers, the room pillars are lost entirely; with 36-foot room-centers, about 55 per cent of the pillars are recovered; and with 39- to 42-foot centers, from 60 to 70 per cent of the pillars are recovered.* Many miners at the present time have not the skill to do the best pillar work, even if the cost were not too high, and for this reason, if a higher percentage of pillar coal is to be won, cutting with machines, which can be operated successfully only on wider pillars, must be employed.

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* Cunningham, F. W., Op. Cit., p. 275, 1910.

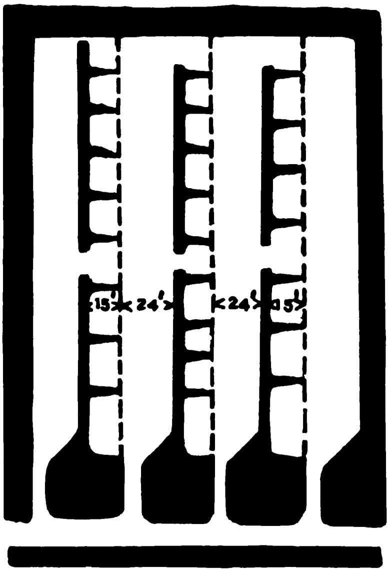

FIG. 13. PILLAR DRAWING, CURTAIN OF COAL

A method adopted for future working at the Marianna and the Hazel mines of the Pittsburgh-Buffalo Company in Pennsylvania and the Annabelle mine of the Four States Coal Company in West Virginia is illustrated by Fig. 14.* The plan as outlined is intended for coal under a cover of 300 to 500 feet. There are two sets of triple main entries separated by an unbroken pillar 50 feet thick. The operation really includes two distinct mines except that the coal goes over the same tipple. The loaded and empty haulage roads on each side are driven 10 feet wide, and the airway is driven 16 feet wide. These widths are necessary in order to avoid the expense of a fourth 10-foot entry.

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* Edwards, J. C., and Gibb, H. M., "An Ideal Method of Mining," Mines and Minerals, Vol. 33, p. 665; and Edwards, J. C., "Machine Mining in Room Pillars," Mines and Minerals, Vol. 34, p. 591.

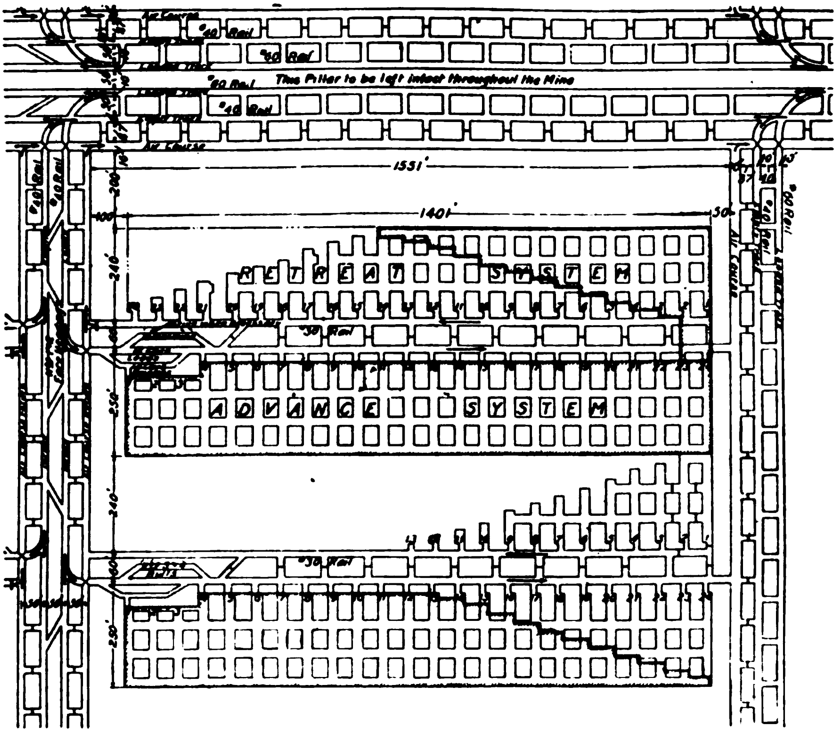

FIG. 14. PROPOSED PLAN FOR PITTSBURGH-BUFFALO COAL COMPANY

The panel system planned is known as the "half advancing and half retreating" system. The panels are divided into blocks 500 feet wide by entries driven "end on" in pairs, and from these butt entries the rooms are turned on the face of the coal. The butt entries on the side toward which the development of the panel is progressing, that is on the inby side, are termed "advance headings"; the exterior or outby entries are "retreat headings." A chain breast machine is used

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*Ibid.

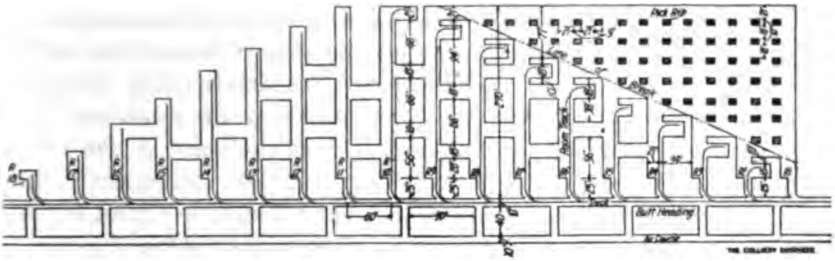

FIG. 15. EXTRACTION OF PILLARS UNDER DRAW SLATE

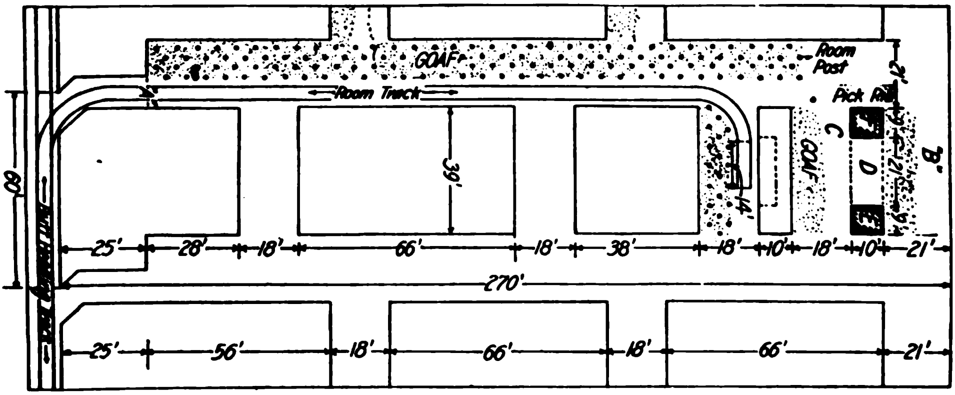

FIG. 16. DETAIL OF PILLAR WORK UNDER DRAW SLATE

in rooms and entries and a short-wall machine on pillars. Both machines continue in use until the last room on the retreat entry is completed; then the short-wall machine is left to finish the pillars, and the breast machine is transferred to another pair of butt entries under development. By the time room 14 is turned, room 2 has been finished, and work can be begun on the pillar between rooms 1 and 2. Rooms on the advance entry are 255 feet long and those on the retreat entry 246 feet long from the entry centers; this difference in length is made, because the chain pillar and entry stumps are brought back with the room pillars on the retreat entries. The method has been worked out for two general conditions; first, where a draw slate is encounted, and secondly, where there is no draw slate.

The method to be used where draw slate is encountered is illustrated by Fig. 15. In this illustration room 1 is shown as finished. In rooms 2 to 9, inclusive, the pillars are being drawn. Room 10 has reached its limit, and the cross-cut at the face is being driven through the pillar to room 9. Rooms 11 to 18, inclusive, are being driven. Fig. 16 shows in detail the method of recovering the pillars by the short-wall machine where draw slate is encountered. From the point *A*, the track is laid in 14-foot sections, and steel ties are used; consequently, the track is easily assembled or detached. Curved rails are used in the same way so as to give easy access to the cross-cuts. After the cross-cut *B* is finished, the curves and two 14-foot sections of the track are detached, and an 18-foot cut is made in the pillar at *C* by working on the butt of the coal and leaving a stump *D*, 10 by 39 feet. The draw slate from the first cut is gobbed in the room proper. The remainder of the draw slate from this cross-cut is gobbed in the outby part of the cross-cut, and the track is laid in the inby part. A cut is next made through the stump *D*, into the gob above, and small blocks or stumps *E* and *F* are left on each side of the cut to be taken out with the pick. After the block *E* has been removed, all the tracks in the cross-cut, except the two curve rails, are removed; when the block *F* has been removed, the curve rails and the two 14-foot sections of straight track are taken up, and the operation of driving through the pillar is repeated as the illustration shows. In this way the room pillar is extracted back to the point *A*.

The method of operation in the second case, where the draw slate is not encountered, is the same as in the first case, except for the manner of attacking the pillars which is illustrated in detail by Fig. 17. After the room has been completed and the cross-cut *B* driven, the two curve rails and seven sections of track are detached; thus the track is left in position to be assembled quickly for easy access to the cross-cut *H*, which is next driven. The 39- by 94-foot pillar is then split from *H* to *B*, and a 9- by 94-foot pillar is left on each side; then an 18-foot cross-cut *D* is driven through the 9- by 94-foot pillar on the rib nearest the gob, and a stump is left to be removed by the pick. The other 9- by 94-foot block is removed in the same way, and the operation is continued until the whole pillar has been removed. The entry stumps and chain pillars of the butt headings are won in the same manner as the room pillars.

Both of these methods are in use at several mines and are meeting with success. In both methods 90 per cent of the coal won is cut by machines. This percentage can be increased considerably, since it has been demonstrated that under favorable conditions part of the pick blocks can be recovered by the machine. One-third of the coal is mined from the rooms and two-thirds from room pillars. When draw slate is encountered, the 39- by 10-foot stump pillar will always afford ample protection to the miners; and where no draw slate is found, the two 9- by 94-foot stump pillars together with the timbering will give adequate protection.

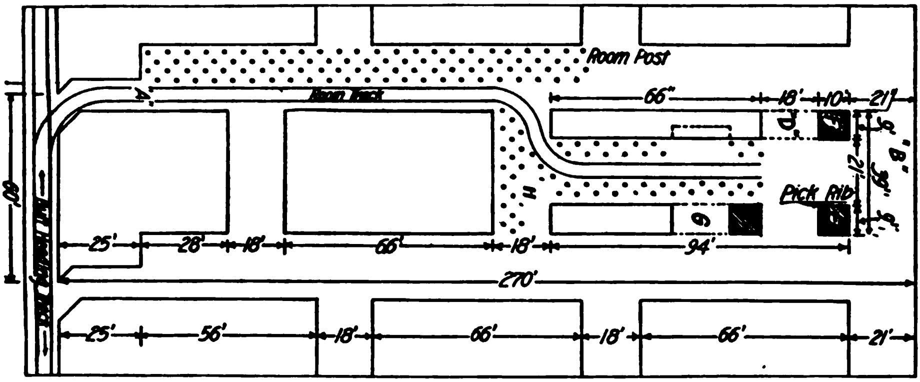

FIG. 17. DETAIL OF PILLAR WORK IN ABSENCE OF DRAW SLATE

Among the plans tried in the Pittsburgh district with the object of reducing the cost of mining by substituting more machine work for pick work is one (Fig. 18) which promised to be successful, but failed, because the miners demanded room-turning prices for the short rooms.* This added cost would have defeated any other advantage of the system. The method, however, seems to be based upon principles which may find application under other circumstances. The method was adopted, because a provision of the mining scale of the Pittsburgh district prohibited the drawing of ribs by machines unless short-wall machines were used. The plan was to continue the employment of the breast machines already in use and thus to increase the percentage of room coal. It seemed profitable to increase the percentage of room coal since the cost for machine-cut run-of-mine coal was 45.28 cents per ton, including a differential of 2/3 of a cent on account of rolls, while the price for pick mining was 64.64 cents per ton; the average price for cutting, loading, and pick work was 51.92 cents. This average was based on the assumption that the working was regular with rooms 25 feet wide on 40-foot centers. Machine work was done with the common breast machines. The mining system was the ordinary method of machine mining with 25-foot rooms on 40-foot centers with cross-cuts three runs wide and room necks 21 feet long. Rooms were 250 feet long, but as the neck was 21 feet long, the length of the actual room was 229 feet. All estimations of the percentage of extraction were based on a block 229 feet long and 120 feet wide. In this old system of mining, 65.7 per cent of the coal was produced as machine coal and 34.3 per cent as pillar coal drawn with the pick. The mine was operated on a run-of-mine basis. The new system was started with 36-foot rooms on 117-foot centers; but it was expected that if the system should prove successful, these dimensions would be changed

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* Affelder, W. L., "Rib Drawing by Machinery," Coal Min. Inst. Amer., p. 282, 1912; and Personal Communication.

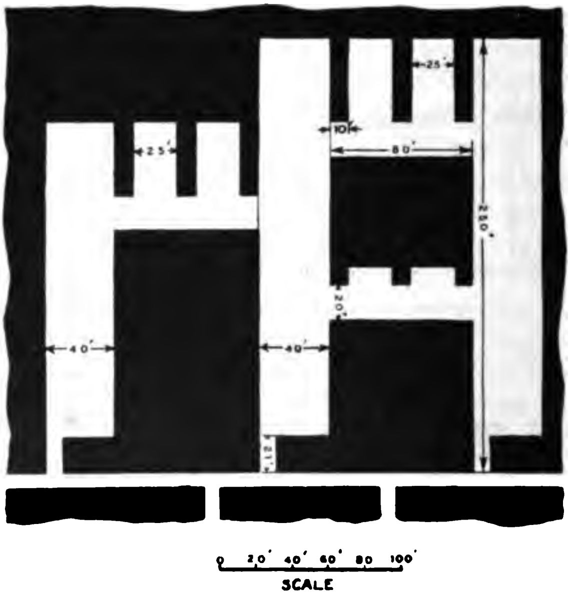

FIG. 18. METHOD OF REDUCING PILLAR WORK IN PITTSBURGH, PA., DISTRICT

to 40-foot rooms on 120-foot centers. The illustration shows the form and dimension of rooms. Two 25-foot rooms were driven from the crosscut with 10-foot pillars on each side. These pillars were somewhat weak, but the rooms were not long, and the pillars were drawn quickly. It was admitted that the system would reduce tonnage for a time if the mines developed were not sufficiently advanced, but it was claimed that when the main rooms had been driven to within 50 feet of their intended length, the production from them would be much greater than from the ribs of three rooms of the older system. As soon as the pillars were drawn, the recovery of the so-called "rooster" coal would rapidly increase the output. In the Pittsburgh bed, the rooster coal lies above the draw slate and a laminated coal 12 inches to 2 feet thick. Although in most mines this coal is not taken out, in the Panhandle district it seems to be better fuel than the bottom coal and is being mined in several places. The cover in the section in which this method was employed was from 75 to 125 feet thick. The small pillars were extracted by pick work, and the rooster coal also was obtained at pick prices.

21. *Connellsville District.*—The methods of production in the Connellsville field have become more intensive than those in the other districts of western Pennsylvania. The excellent quality of the Connellsville coke and the fact that the coal from which it is produced is found in only a limited area have made the coal so valuable that it has been found advisable to pay particular attention to high percentage of extraction. In this district, there has been, moreover, no objection to the crushed coal from pillars as the product goes to the coke ovens where fine coal is desirable. The same fact influenced the relative sizes of rooms and pillars. While in the gas coal district it was thought desirable to take as great a quantity as possible from rooms, in the Connellsville district the practice of getting a large part of the coal from the pillars has developed in order that the percentage of extraction may be as high as possible. It is of interest to trace the more recent developments in mining practice here, because the extraction in the better planned mines of this district is unusually high.

Conditions and methods in this district are discussed by F. C. Keighley* as follows:

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* Keighley, F. C., "Mining Coal with Friable Roof and Soft Floor," W. Va. Coal Min. Inst., Dec. 10, 1914; Coal Age, Vol. 7, p. 1008.

"The coking coal known variously as the No. 8, Pittsburgh, or Connellsville seam has in places an extremely bad roof. This difficulty is strongly marked in the Connellsville basin, but can be found in other troughs. . . . The thickness of the coal and its softness might lead one to anticipate that it could be mined cheaply, but the friable roof creates a difficulty which its other qualities cannot overbalance. . . .

. . . "I have projected workings at depths ranging from 200 to 700 feet, using a dozen or more different schemes, and have never found any marked difficulty in protecting the main headings and air-courses of any mine. But I have experienced some trouble occasionally in protecting the branch, or butt, headings, and I have always had more or less trouble with the rib coal in the rooms.

"It is true that in many cases the roof will fall in headings and air-courses in spite of all that can be done, but such falls are gradual and are removed as part of the regular mine operations. On the other hand, when falls occur in the rooms they often come suddenly and cover a large area, and the break extends so far above the coal that they often reduce and may entirely cut off the production from a certain section of the mine, not only for a day, but perhaps for weeks at a time.

"It is clear, then, that any improved method of mining must provide for the protection of the rooms rather than for the care of the headings. . . .

"Panels have been projected 1,000 feet wide and 2,000 to 4,000 feet long. . . . Such a panel is subdivided into a number of smaller panels that are themselves served by two parallel entries driven at right angles or at some other angle to the flat or face heading depending on the pitch of the coal. These are known as butt headings. These sub-panels are generally 1,000 feet long and 300 to 600 feet wide. . . .

"Various widths have been chosen for rooms with 8 to 30 feet as limits, but the general belief is that rooms and headings should be driven 10 feet wide in the Connellsville region. There seems to be no opportunity to improve on this width. The best work has been obtained with large room ribs—50, 70 and 90 feet thick; but the success has not been as great as might be expected, though, with room ribs of the two larger dimensions, and with any ordinary care in mining, a general creep or squeeze cannot occur.

"This is not true when room ribs are made of smaller dimensions, such as 30, 40 or even 50 feet. In the initial stage of rib drawing with such light ribs great success is secured, but when trouble occurs it is usually in the form of a general squeeze or creep that almost paralyzes the output. It has often seemed for a time that the small ribs in the rooms resulted in cheaper mining; but when a squeeze or creep took place the small rib did not permit of the driving of a new road with safety and profit, and consequently the coal remaining in the rib could not be taken out.

"With 70-, 80-, or 90-foot ribs there is always sufficient coal left to permit driving a new road with safety through the pillar no matter how badly the roof may have fallen or the coal be shattered on the edges of the pillars.

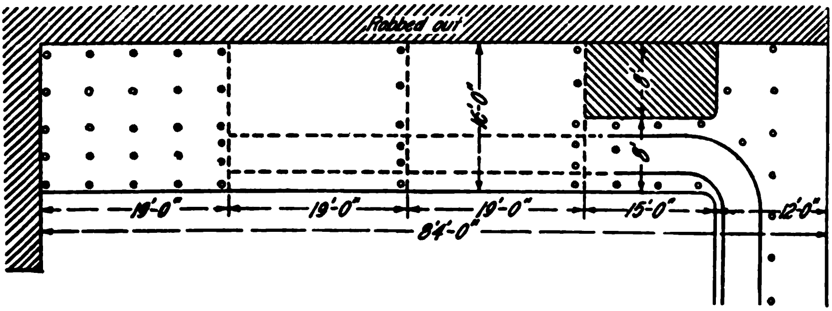

FIG. 19. PILLAR DRAWING IN CONNELLSVILLE, PA., DISTRICT

"Nearly all experienced miners concede that with narrow ribs only 50 per cent of the coal is recovered. The best results claimed is 65 per cent, while 90 and 95 per cent has often been recovered with the larger-rib system. The problem is whether the heavy cost of timber and the still greater cost of labor will counterbalance the loss of from 35 to 50 per cent of coal. I am disposed to believe that the larger rib, making a larger yield possible, will assure a handsome margin."

Fig. 19 illustrates a modern method of pillar drawing.* In this method a cut is made across the pillar, and an 8-foot stump is left; then this stump is taken in a retreating direction in from two to four sections, according to the width of the pillar. In the example given the rooms are 12 feet wide and are on 84-foot centers. After the removal of the coal in each section, the props are drawn and the roof is allowed to fall, the break being controlled by a row of props across the end of the cut. After the last section of the stump has been taken out, the last of the track drawn, and the roof dropped across the room on the line of the end of the remaining pillar, another cut is made across the pillar and the process continues. While the falls represented in the sketches appear to be large, a better break is obtained with these than with short falls. This method has proved to be safer for the miners and to give a greater recovery of posts and coal than the methods which preceded it.

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* Cunningham, F. W., "The Best Method of Removing Coal Pillars," Coal Min. Inst. Amer., p. 275, 1910; p. 85, 1911.

A highly developed and systematized room-and-pillar method is the so-called concentration method used in some of the mines of the H. C. Frick Coke Company,* and developed largely by Patrick Mullen, one of the company's inspectors. This method was designed to satisfy certain requirements, among which were safety of operation, completeness of extraction, reduction of cost through the greater use of machines, and an increase of daily output per man. A patent covering this method has been applied for.

It is well understood that liability to accidents is decreased by close supervision. Under the older system in the Connellsville region, it was possible for the face boss to visit each working place only once in two or three days. In order to increase the amount of supervision without increasing the number of officials, the plan of getting the working faces closer together was tried; this plan, however, necessitated a decrease in the number of working places and in the number of workmen, and it was realized that only by increasing the production of each miner could the output of the mine be kept up.

The only possible way of increasing the output per man was by replacing pick work with machine work. This substitution was made in room work; but it was found that, on account of narrow headings and narrow rooms with large room centers, machines in the narrow work alone would not accomplish the desired results, since the bulk of the coal comes from the pillars. The problem of the use of machines for pillar extraction, which was an entirely new one in the Connellsville district, has been worked out very successfully (see Fig. 20).

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*Mullen, Patrick, "New Mining Methods as Practiced by the H. C. Frick Coke Company," Proc. Engrs. Soc. W. Pa., Vol. 32, p. 714, 1916; Coal Age, Vol. 10, p. 700, 1916; Howarth, W. H., "Mining by Concentration Method," Coal Min. Inst. Amer., Dec. 22, 1916; Coal Age, Vol. 9, p. 125, 1916.

The mine is blocked by driving at *A* double butt entries, 10 feet wide on 50-foot centers and 1,200 feet long, across the panel with breakthroughs every 100 feet; the pairs of butt entries being driven 350 feet apart, the panel is divided into blocks about 350 by 1,200 feet. These blocks are then subdivided into blocks about 90 by 100 feet, by 12-foot face rooms at *B* 350 feet long on 112-foot centers, driven at right angles to the butt entries and connected by 10-foot breakthroughs on 100-foot centers. A pillar of this size is considered ample to support any thickness of cover under any conditions of floor or cover to be found in the Connellsville region. In this manner a whole panel can be prepared for the intensive part of the work in which butt rooms are driven from the face rooms 10 feet wide on 25-foot centers.

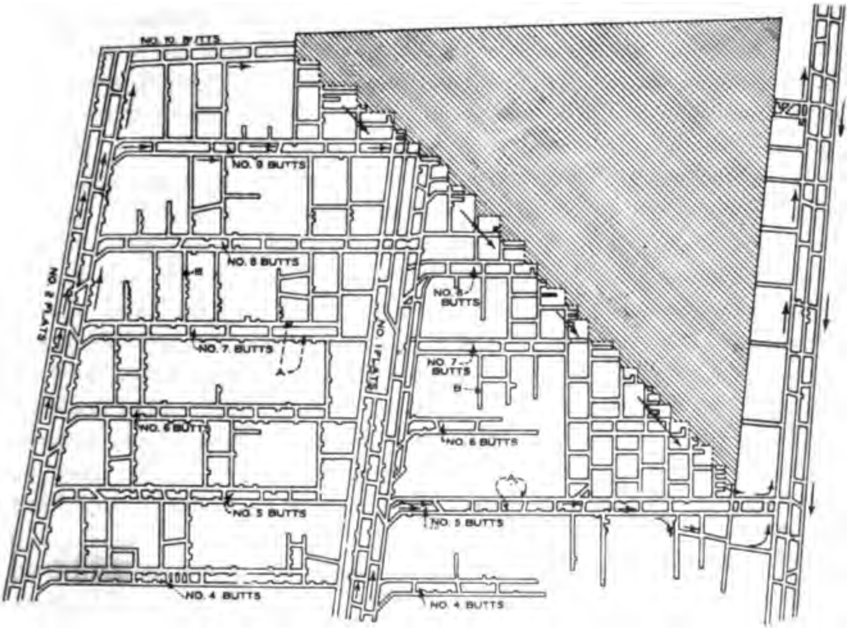


FIG. 20. CONCENTRATION METHOD IN CONNELLSVILLE, PA., DISTRICT

As the main face room advances, the necks of the butt rooms to be driven are excavated to a depth of three machine cuts. After a main face room has been advanced 50 feet, there are available for the machine to cut two places which allow a production of forty tons; and when the room has advanced to a point where the first crosscut is turned off, there are three places to cut in each main face room yielding sixty tons. This main room may continue to the end of the section or to the end of the coal field, butts or producing entries being turned off at projected distances. It is necessary that the operation be carefully planned and that the proper order of the work be closely adhered to. Fig. 21 shows the general schedule of operations with the position of the line of roof fracture at different dates. It is claimed that the general plan can be easily modified to suit all conditions such as depth of cover, presence or absence of draw slate, and nature of coal, bottom, and roof.

The projection takes three forms known as (a) maximum, (b) medium, and (c) minimum, according to the rate at which coal is produced (see Fig. 22.) The maximum plan is applicable where the thickness of cover does not exceed 125 feet, where the coal is hard, and where the general physical conditions of roof and bottom are good. The medium plan is applicable where the cover does not exceed 250 feet with the same physical conditions of the coal, bottom, and roof as for the maximum plan. The minimum plan may be applied to coal underlying any thickness of cover; the coal may be hard or soft, and the physical conditions of roof and bottom may be good or bad, provided, of course, that mining machines in any form can be used.

With the minimum plan, the butt rooms are driven in succession so that each room is 50 feet beyond the one succeeding. Two butt rooms advancing furnish 40 tons and one butt rib retreating furnishes 40 tons, or a total of 80 tons on the retreat; the main face room advancing yields 60 tons, or a total of 140 tons from one main face room. These quantities apply, of course, to coal of the thickness of that mined in the Connellsville basin — about 7 feet.

The medium plan will yield the same tonnage from the advancing main rooms, but the retreating work is so arranged that the face of each butt room is 30 feet behind that of the preceding room. This arrangement allows three butt rooms to be advanced at a time with a production of 60 tons, while two butt ribs are being extracted with a production of 80 tons; thus 140 tons are taken from the butt rooms and ribs and 60 tons from the advancing main rooms, a total of 200 tons for each main room.

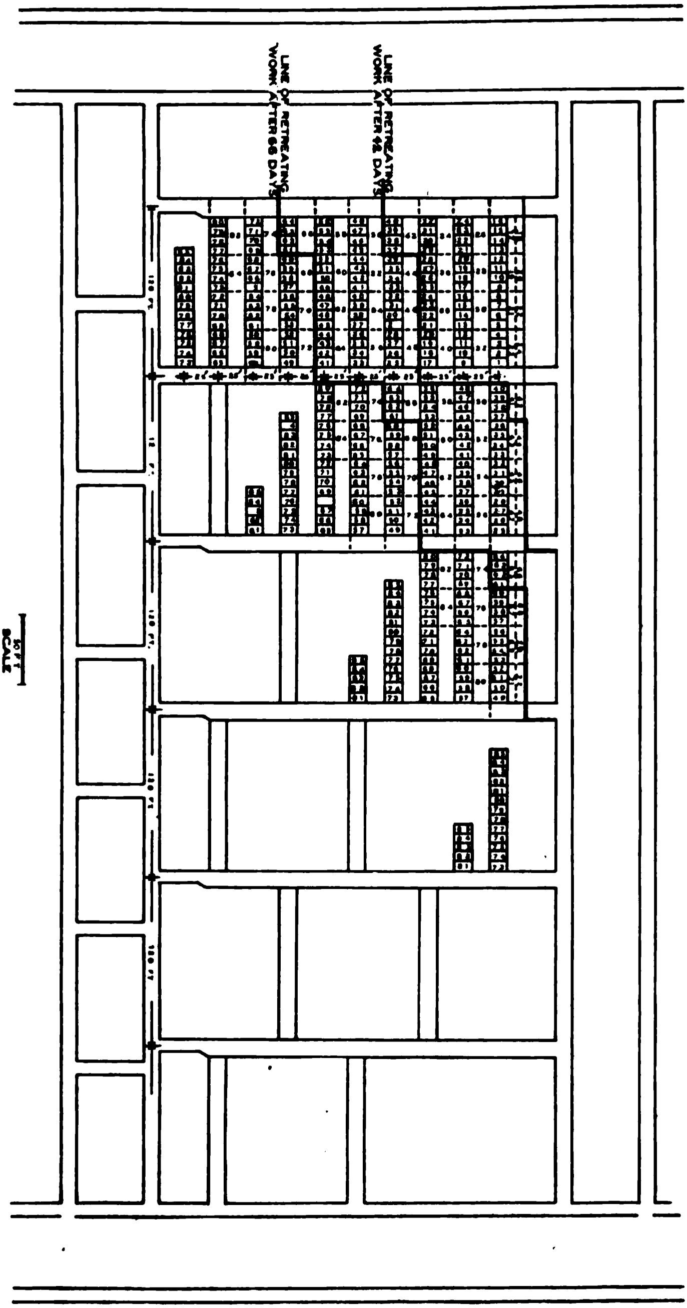
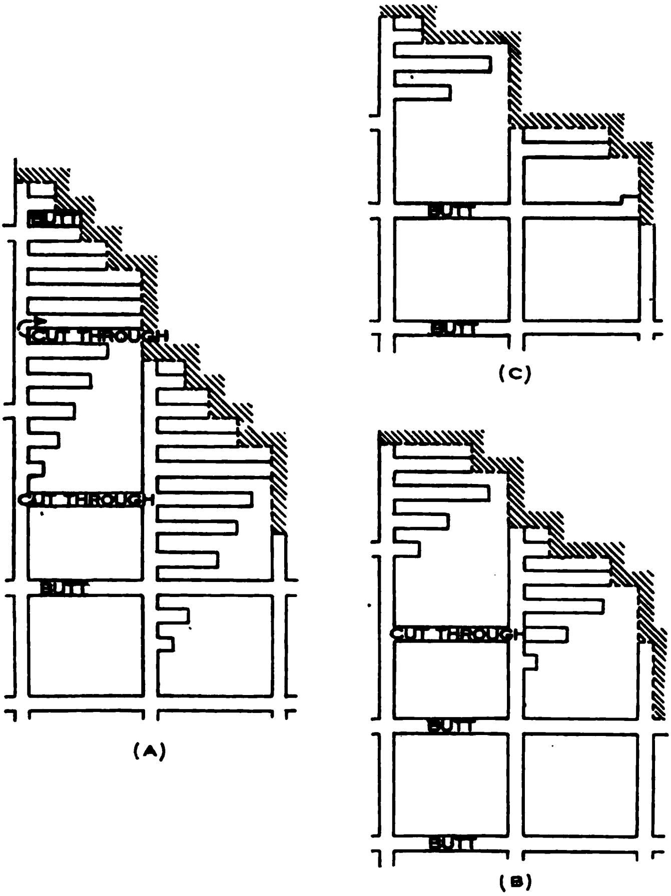

LINE OF RETREATING WORK AFTER 68 DAYS
LINE OF RETREATING WORK AFTER 42 DAYS
120 FT.
120 FT.
120 FT.
120 FT.
120 FT.
10 FT
SCALE

FIG. 21. CONCENTRATION METHOD IN CONNELLSVILLE, PA., DISTRICT—ORDER OF WORKING

In the maximum plan the working of the butt rooms is so timed that the face of one room is 15 feet behind the face of the preceding one; four butt rooms are advanced and four butt ribs are simultaneously withdrawn. The four advancing butt rooms will produce 80 tons and the four retreating butt ribs will produce 160 tons. With the 60 tons produced from the advanced main room, there is thus produced 300 tons for each main room.



(A) (C) (B)

FIG. 22. CONCENTRATION METHOD—MAXIMUM, MEDIUM, AND MINIMUM PLANS

The work is thoroughly systematized and proceeds with great regularity. After the miner has cleaned up his place and the day's run is completed, the machine crew enters and cuts the place to a depth of approximately 7 feet. Following the machine crew, the timber men reset any posts which the machine men have removed, post up any cross bars which have been notched in the coal over the machine cut, and put the place in good condition according to a prescribed system of timbering. The timber men are followed by the driller who bores the holes with an electrically driven drill. The driller is followed by the shot firer who charges and tamps the hole, and after an examination of the conditions, fires the charge. After the coal has been shot down, empty cars are placed by the gathering locomotives so that when the loader arrives at his working place in the morning he finds it in safe condition, the coal ready to load, and the empties in place. Miners loading under these conditions regularly obtain 18 to 20 tons per shift. The average of the loaders for short-wall mining machines in all mines of the company for the month of August, 1916, was approximately nineteen tons per shift. At mines where there is a full equipment of mining machines, the machine coal runs from 80 to 90 per cent of the total output. The recovery under the concentration system is from 90 to 92 per cent, while under the ordinary methods it is 80 to 85 per cent.* In the values given, the top or bottom coal left in place is not considered. The average thickness of top coal left is about 6 inches and the values for extraction, based on the entire thickness of the seam would be somewhat lower than the values given. Coal is left for two reasons. In the entries, from 6 to 8 inches of top coal is left as a protection. In the room work, such top or bottom coal is left in place as is necessary to keep the sulphur content of the coke made from the coal down to the required amount. It is found that the highest sulphur content of the bed occurs at the top or at the bottom and, by frequent analyses, it is determined how much of this top and bottom coal may be left.

22. *Central Pennsylvania.*—A method known locally as the "Big Pillar System" has been developed to meet conditions incident to the soft bottom in the Lower Kittanning, "B," or Miller, bed in the southern and eastern parts of Cambria County, and in the adjoining territory.†

The physical conditions for which this system was developed include a hard roof, very difficult to break, and a soft fire clay bottom. A sand rock from 10 to 40 feet thick occurs above the coal; but between this sand rock and the coal there is usually from 1 to 6 feet of slate or sandy shale, which is more or less affected by the air and which breaks away from the sand rock, especially in the roadways. The falling of this top tends to relieve the pressure, but not enough to prevent squeezing. The worst squeezes are encountered where the sandstone is only 10 feet thick.

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*Dawson, T. W., Personal Communication.

†Silliman, W. A., "Big Pillar System of Mining," Proc. Coal Min. Inst. Amer., p. 76, 1911.

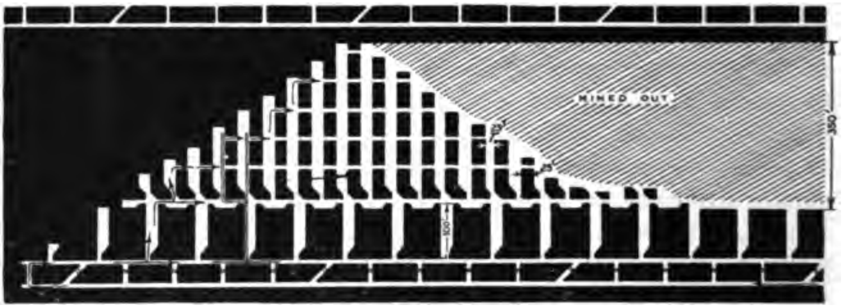

FIG. 23. "BIG PILLAR" METHOD USED IN CAMBRIA COUNTY, PA.

Under these conditions, the ordinary system of turning rooms with 40-, 50-, or 60-foot centers does not work satisfactorily. When the room pillars are drawn back to the stump, under the ordinary system the pressure is so great that a stump, even 30 or 40 feet square, will not protect the entry. Instead of breaks occurring in the roof along the line of the stumps the bottom breaks and heaves, and squeezes occur. Since the coal is soft and has a columnar fracture, the stump is badly crushed, and no amount of timbering is sufficient to prevent the closing of the entry. Thousands of feet of entry and much coal have been lost as the result of squeezes in a bed of this kind. The bed is only about 3½ feet thick so that it is necessary to take up the bottom, and the provision of space for storing bottom is one of the considerations involved in planning this system. The average dip is about eight per cent.

In the "Big Pillar" method (Fig. 23), haulage entries are driven on the strike and rooms are turned up the pitch. Entries are 10 or 12 feet wide. Rooms are turned on 100-foot centers from the entries, and, at a distance of from 100 to 125 feet from the entry, rooms called "crooked" rooms are turned at right angles, that is, parallel with the entry. There is thus left along the side of the entry a series of blocks 75 by 75 feet or 75 by 100 feet, according to the length of the rooms driven from the entry. As soon as a crooked room has intersected the straight room toward which it is being driven, an intermediate room is turned up the pitch from the crooked room; thus the rooms above the crooked rooms have 50-foot centers. The straight rooms are driven to such distances that the roof will break at the edge of the big pillar or by settling will relieve the strain. Sometimes they are only 250 feet long, and sometimes, under heavy cover, as much as 400 feet.

When the straight rooms are started, they are widened on the outby side so that the cross, or crooked, room can be turned off the straight rib, a matter of importance because of the necessity of storing bottom which is taken up in the roadway. Beyond the crooked rooms, the straight rooms are widened on the inby side; thus the men who drive a room are able to start the drawing of the pillar as soon as the room is finished. The room pillars are drawn back to the crooked rooms, and the irregular little blocks caused by the necks of these rooms are removed as completely as possible. The big block is then left standing to serve as a barrier to protect the entry, and if the space mined out is sufficiently broad, the roof will usually break. Even if the roof does not break, the strain seems to be relieved before reaching the entry. The upper edge of the big pillar may be badly crushed, and the roadway of the room may be heaved almost down to the entry, but the entry itself will be practically unaffected.

When the entry is finished and the stumps are being drawn, the system presents a special advantage in that a better output can be obtained than with the smaller stumps. Where stumps are small, the output is limited to the work of two gangs, but with the big pillars eight or ten places may be worked at all times on the retreat. The big pillars are split by a room driven up from the entry at the same time that a skip is taken along the rib of the old room. These two working places are cut through to the old falls about the same time, and the intervening portions of the pillars are brought back. This method leads to large recovery of coal, although there are no statements available concerning the exact percentage. There is some loss in the extraction of the pillars, and the coal at the edge of the big pillars is badly crushed. The method is not used in other beds in the same district, because the conditions are better.

In the Somerset County district, there has been developed a panel system which permits a high degree of concentration of work and a large percentage of extraction. The coal is low, and the miner is obliged to push his cars to the face of the room and to drop them down to the entry. The seam often dips from two to five per cent, and butt entries are driven off the main haulage slope on a grade of one per cent in favor of the load. From these, entries are driven to the rise at convenient distances, from which rooms are turned on the strike of the bed. Not only does the method result in a high percentage of extraction and facilitate the handling of the cars by hand in the rooms, but it also concentrates the work of mining. Two men working together will produce from ten to twelve tons of pick-mined coal per day, but when the men work singly it has been found that a good miner can load from seven to eight tons per day.[*] Under this system, the total extraction is reported to be about 93 per cent. About 50 per cent of the coal comes from rooms and entries, and the remainder from pillars.[†] This method is similar to that illustrated in Fig. 32, page 102, which shows a plan of operation of the Carbon Coal Company in West Virginia.

Because of the recognized objections to the room-and-pillar system, much attention has been given to the possibility of employing the long-wall system in the Pittsburgh bed and in other beds of western Pennsylvania. So far as can be learned, there is only one mine at which a long-wall system is being used in these districts, although there is an approach to it in some so-called "panel long-wall" or "block long-wall" methods. In these methods, dependence, however, is not placed on the weight to break down the coal; in fact, weight at the face is prevented so far as possible by causing the roof to break near the face.

It is worth while to review some of the experiments in the introduction of long-wall methods, because it is only by these methods that complete extraction is attained, although there is a close approach to it in the best applications of some forms of pillar working.

One of the attempts[‡] was started about 1899 in the Lower Kittanning, "B," or Miller, bed where the coal was from 3 feet, 6 inches to 3 feet, 10 inches in thickness, and had an average dip of eight per cent. The roof was of blue slate and the floor of hard fire clay. Most of the coal lay under a fairly level surface about 174 feet thick, near the top of which was a moderately hard sandstone. The plan adopted involved blocking out the mine with entries and taking out the coal in each entry on the retreat. The faces were 250 to 300 feet

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[*] Majer, John, "Mining by Concentration Methods," Coal Age, Vol. 9, p. 845, 1916.

[†] Coxe, Edward H., Personal Communication.

[‡] Claghorn, Clarence R., "A Modified Long-wall System," Mines and Minerals, Vol. 22, p. 16; Thomas, J. I., "Mechanical Conveyors as Applied to Long-wall Mining," Proc. Coal Min. Inst. Amer., p. 55, 1907; Delano, Warren, Personal Communication.

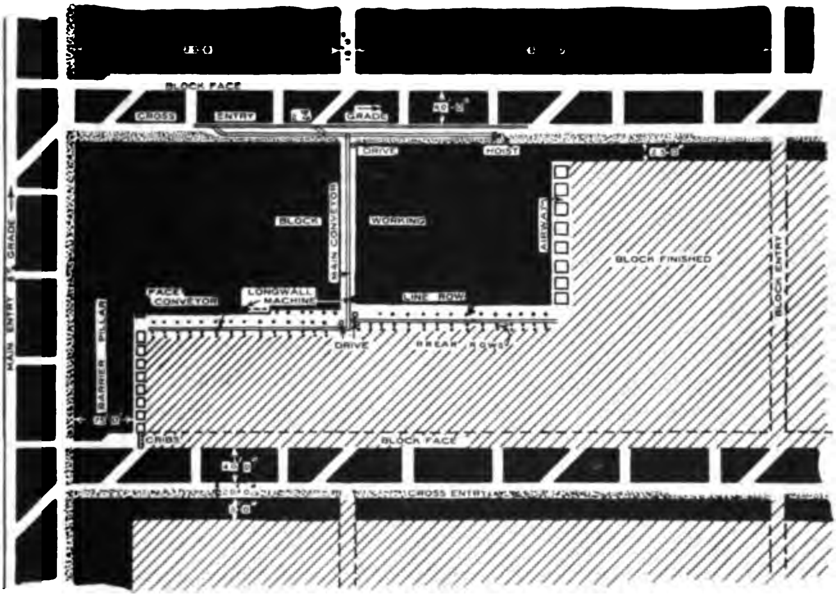

FIG. 24. BLOCK LONG-WALL WITH FACE CONVEYORS

long. One machine-cut produced from 125 to 150 tons of coal. Break-rows were made of stout, round, hardwood posts, 6 to 8 inches in diameter. These were capped with a 2-inch lid of soft wood set on a little slack to facilitate drawing. It was found that the track along the face took up too much space, and a conveyor, which was put in made it possible to set the props closer to the face. At first there were two faces, one slightly in advance of the other, each served by a conveyor which delivered coal to cars let down the block entry. Later, as shown by Fig. 24, a conveyor, by which the coal was lowered to the cars on the level, was installed in the block. Because of the unfavorable trade conditions in 1907, operations could not be carried on with the regularity essential to the success of the system as then used. The attempt, accordingly, was temporarily abandoned, but a revival of the plan is being seriously considered.

Another attempt at long-wall mining was made with very similar mechanical arrangements in the Cement seam near Johnstown. In this instance two faces were cut, but a shortage of power compelled the abandonment of the experiment before the second face had been completed. Until that time the mining had been economical and the recovery was almost perfect.*

At present the Maryland Coal Company of Pennsylvania is using eight long-wall conveyors at St. Michael. The coal is about forty-two inches thick. No description of the operation is available, but it is evidently considered successful, as the number of conveyors is being increased. Two other companies in Pennsylvania and one in Maryland have recently decided to employ the same method.†

23. *Summary of Facts Relating to the Percentage of Recovery in Pennsylvania.*—The recovery in the Pittsburgh bed, not including the coke district, is estimated to be about 80 per cent. This estimate is made on the following basis: The actual tonnage mined is compared with the computed tonnage of the district worked out; everything between the fire clay and the drawslate is included, and no deduction is made for impurities in the bed or for the average thickness of four inches left on the bottom. This computation is obtained from one of the largest operating companies of the district and is based upon actual measurements. It is the opinion of this company, assuming that this method of calculation is used, that 85 per cent is probably the best possible recovery in this district. Some other companies claim an extraction of 90 to 95 per cent, but this is calculated after deducting the coal left in the bottom, the bearing-in bands, and any other impurities in the bed which are taken out and not weighed. Such high recoveries, of course, imply careful planning for the extraction of pillars and for the execution of this work without delay.‡

Another company the workings of which lie along the Monongahela River south of Pittsburgh, estimates the recovery as 86.7 to 90.6 per cent.

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* Moore, M. G., Personal Communication.
† Link-Belt Company, Personal Communication.
‡ Schluederberg, G. W., Personal Communication.

In the Connellsville district the best practice gives from 80 to 85 per cent with the methods ordinarily used there. With the new "concentration" method of the H. C. Frick Coke Company a recovery of from 90 to 92 per cent is obtained.*

In the Johnstown district it seems impossible to obtain estimates of the percentage of extraction, because all the seams in that district vary in thickness within short distances, and are somewhat cut up by rolls. In some seams, for example, the Miller seam in the vicinity of South Fork, the recovery is almost perfect. The conditions are favorable, the roof being well adapted to extraction of pillar stumps in retreating.†

One of the companies operating in Jefferson County claims an extraction of 90 per cent. The operations are in the Lower Freeport bed, and conditions are somewhat peculiar because of bad roof, lack of uniformity of the seam, and faults. Each district requires individual development before an estimate can be made of the proportion of faults to the whole area, and it is impossible to make an accurate estimate of recovery until a district has been completely worked out. The value given represents the proportion of coal extracted from the area mined in which coal existed, and does not apply to the area of faults.‡

One operator in Clearfield County estimates an extraction of 95 per cent, based upon the amount of coal mined up to December, 1916.¶

One of the companies operating in Somerset County estimates that, where mines are operated in an area of less than 300 acres and under a cover not exceeding 200 feet, the recovery should be, and in a number of instances is, in excess of 90 per cent. In the case of a property of 1,000 or more acres, where the coal extends underneath a hill giving cover of 300 to 700 feet, the recovery is from 85 to 90 per cent. Low coal, faults, and adverse grades still further reduce this percentage.§

The attainment of the higher percentages in Pennsylvania has been reached only within very recent years, and is not yet by any means universal. There are still in operation a large number of old mines, mostly small, in which high percentages of extraction are not obtained. The more recent operations are planned for, and give, probably as high a yield of coal as can be expected from the area worked.

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* Dawson, T. W., Personal Communication.

† Moore, H. G., Personal Communication.

‡ Van Horn, H. M., Personal Communication.

¶ Personal Communication.

§ Delaney, E. A., Personal Communication.

24. *Maryland.*—In the Georges Creek region of Maryland, the Big-vein coal has been mined for about one hundred years, and the methods used there furnish an illustration of progress in coal mining engineering which is especially interesting in view of the increasing attention given to the percentage of recovery.

In a paper on maximum recovery of coal,* H. V. Hesse discussed the wasteful early methods of mining in the Georges Creek region and suggested economic methods, as follows:

> "A region of uniform and unusually severe conditions in the bituminous fields has been selected to illustrate the results obtained over a long period. The Georges Creek region of Maryland, with remarkable deposit of semi-bituminous 'Big-vein' coal, has operated in this seam and shipped to the market for nigh unto a hundred years. . . . More than one miner still lives who 'dug coal' before the war with the South, and . . . he tells of the detail method of extracting the coal, on account of which thousands of tons lie buried to-day, much beyond recovery. . . .
>
> "The 'Big-vein' seam occupies the geologic horizon of the Pittsburgh bed, but differs considerably in structure and quality from the coal of Pittsburgh, Connellsville, and Fairmont. . . . The top coal averaging 2 feet thick is left up for a roof. Where this comes down the strata immediately above promptly follows. Very little of this top coal is therefore recovered. Both roof and breast of the seam contain slips known among the miners as 'horsebacks,' which frequently fall out without any warning. The coal is soft and the 'butts' and 'faces' entirely absent.
>
> "The methods of extraction in vogue at different periods in the history of this field have established the fact that it is impossible to maintain wide working places for any length of time. Headings are driven 8 feet wide and rooms from 12 to 15 feet. In the earlier days there was practically no definite system of extraction, headings and rooms being driven at random and no pillars recovered. Fig. 25 shows such a method in use about 1850. This is reproduced from an actual survey made under the most trying circumstances. . . . It is estimated that fully 55 per cent of the original coal, not counting the top coal, remains and it is expected to recover at least one-half of this, or 27 per cent of the original, by careful operation and the use of about double the amount of timber necessary under a good system of mining. The maximum cover over

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* Hesse, H. V., "Maximum Recovery of Coal," Proc. W. Va. Coal Min. Inst., p. 75, 1908; and Mines and Minerals, Vol. 29, p. 373.

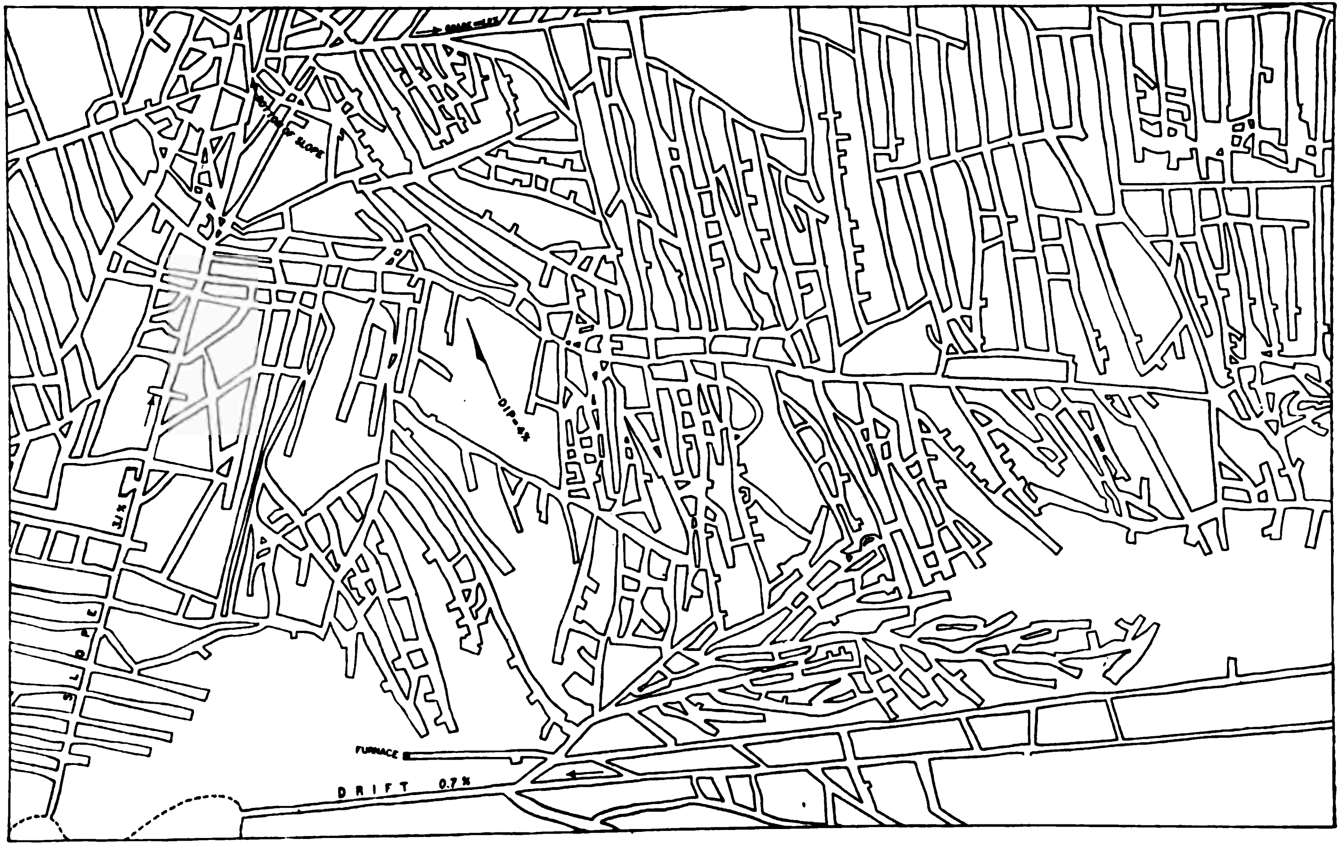

FIG. 25. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1850

this district is 300 feet and the few comparatively large pillars, which were inadvertently left standing at irregular intervals, saved the balance from being crushed. In other sections of the same mine where a similar method was followed, but these large pillars not left in, the workings are entirely closed and the remaining pillars, containing over 50 per cent of the original coal, probably lost forever. Fortunately mining operations during this period were not conducted on a large scale and, consequently, the territory thus affected is limited to a very small portion of the company's holdings.

Fig. 26 "illustrates two methods followed during the years between 1870 and 1880. These workings are inaccessible to surveys at the present time owing to the creeps and squeezes induced by the irregular method of robbing the small pillars. . . . In the first method . . . the rooms were 14 feet wide and pillars 26 feet. These pillars were found to be totally inadequate and extracting them impossible. Cross-cutting the pillars at frequent intervals was then attempted after completion of the rooms, but this was generally accompanied by creeps closing a whole district at a time. The maximum height of the superincumbent strata in this territory is 200 feet.

"The second method shown on Fig. 26 was adopted later. . . . By this method headings were driven from the main entry on the rise of the seam at intervals of 1,000 feet to the level above, and two pairs of cross-headings turned to the right. The rooms were driven from these cross-headings at 50-foot intervals and 14 feet wide, leaving a pillar of 36 feet. The length of the rooms varied from 300 feet to 550 feet. These pillars were also of insufficient size, robbing was conducted spasmodically and although more coal was recovered than in the adjoining districts a great deal was lost. In addition to the small pillars, the method of robbing them was calculated to promote squeezes. It appears to have been the method to hold the strata with props until sufficient coal had been removed to enable the weight to break the props. As a general rule, however, before this was attained the weight had induced a creep which is well known to have no limits within a territory of small pillars.

Fig. 27 "represents a method in use in 1890. . . . Rooms were turned as shown from all headings on 100-foot centers and pillars split by half rooms. The length of rooms varied from 300 feet to 600 feet and they were 14 feet wide, leaving pillars 42½ feet wide. These pillars were not strong enough to support the overlying strata of 500 feet and the usual creep resulted when pillar drawing commenced. . . .

Fig. 28 "shows a method adopted in 1900. The maximum dip is 15 per cent and the greatest thickness of superincumbent strata 425 feet. The slope, together with parallel air-course and manway, are sunk on the heaviest dip of the coal and double entries turned off to right and left at intervals of 1,000 feet on grades of 1¼ per cent to 2¼ per cent in favor of the loads. From these haulways, cross_headings are deflected at intervals of 240 feet at an angle of about 25 ,degrees and driven on a grade of 4 per cent to 7 per cent. Rooms varying in length from 100 to 800 feet are turned on the rise of the coal from

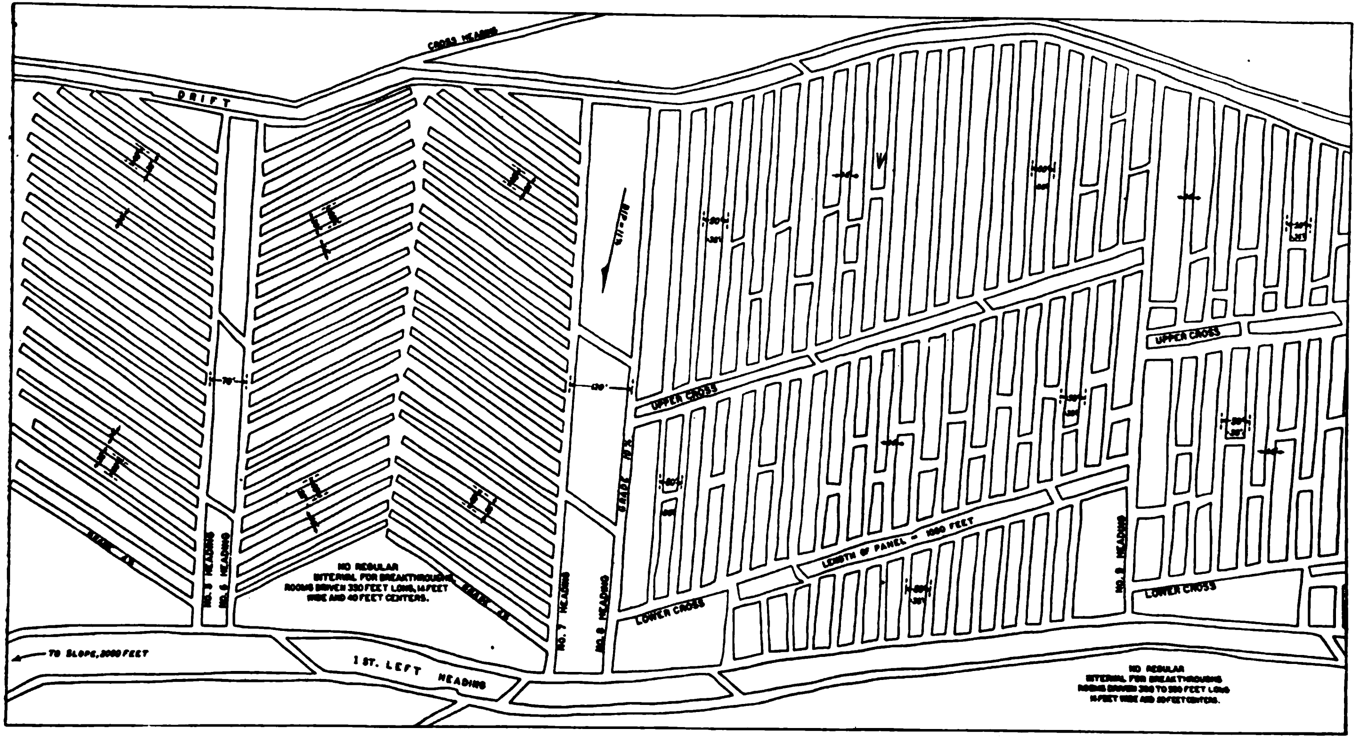
CROSS HEADING
DRIFT
TO SLOPE, 3000 FEET
1 ST. LEFT HEADING
NO. 5 HEADING
NO. 6 HEADING
NO. 7 HEADING
NO. 8 HEADING
NO. 9 HEADING
70'
130'
DIP=11%
GRADE 1½%
UPPER CROSS
LOWER CROSS
LENGTH OF PANEL = 1000 FEET
NO REGULAR
INTERVAL FOR BREAKTHROUGHS.
ROOMS DRIVEN 330 FEET LONG, 14 FEET
WIDE AND 40 FEET CENTERS.
NO REGULAR
INTERVAL FOR BREAKTHROUGHS
ROOMS DRIVEN 300 TO 350 FEET LONG
14 FEET WIDE AND 20 FEET CENTERS.

FIG. 26. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1870–1880

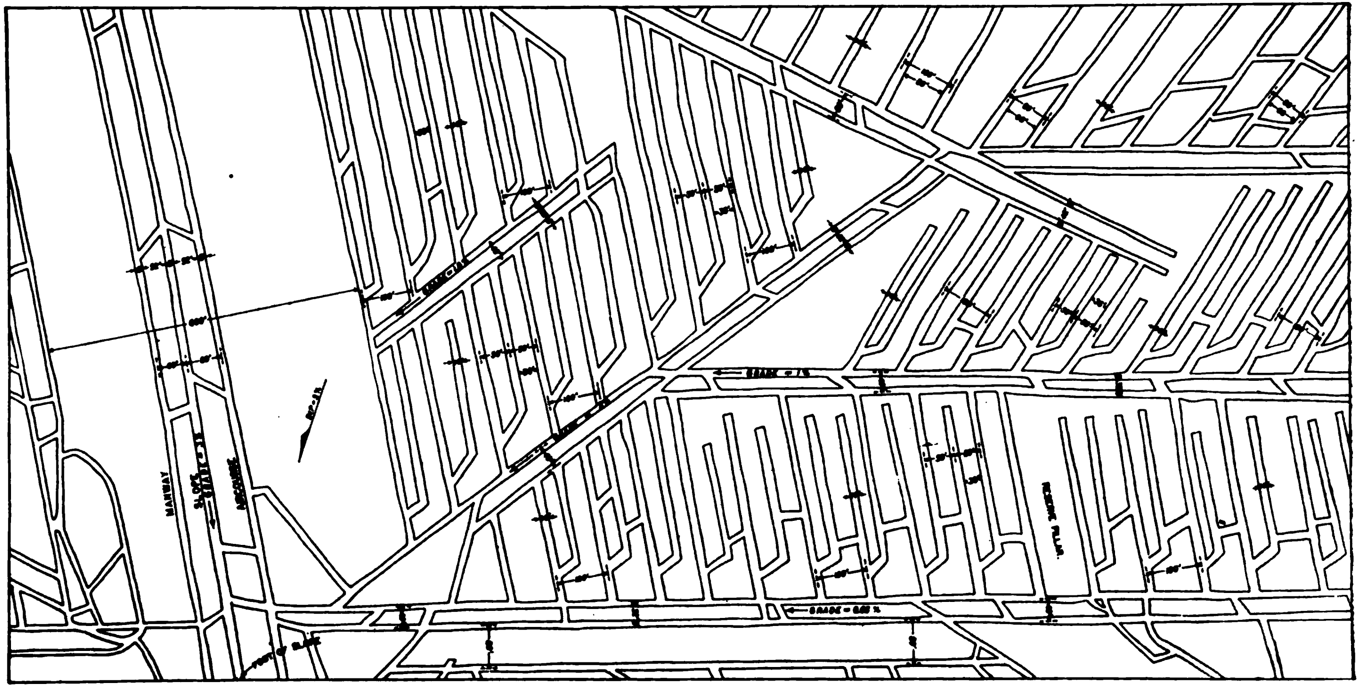

FIG. 27. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1890

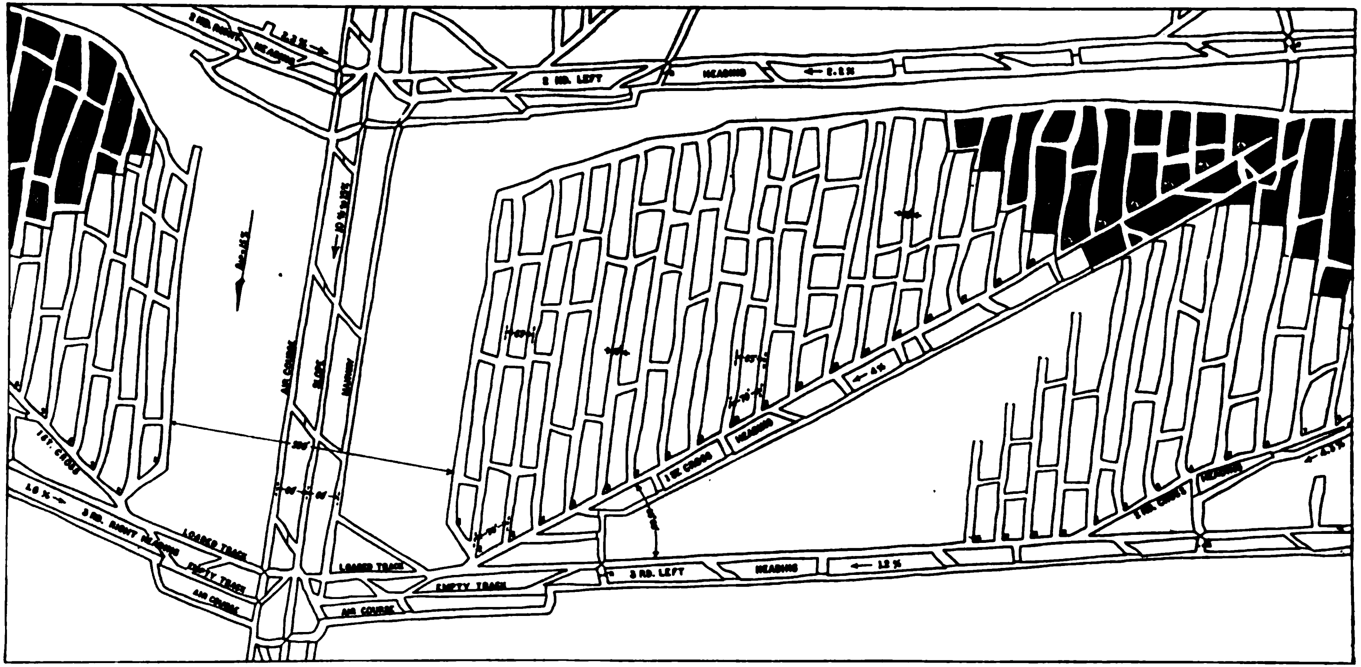

FIG. 28. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1900

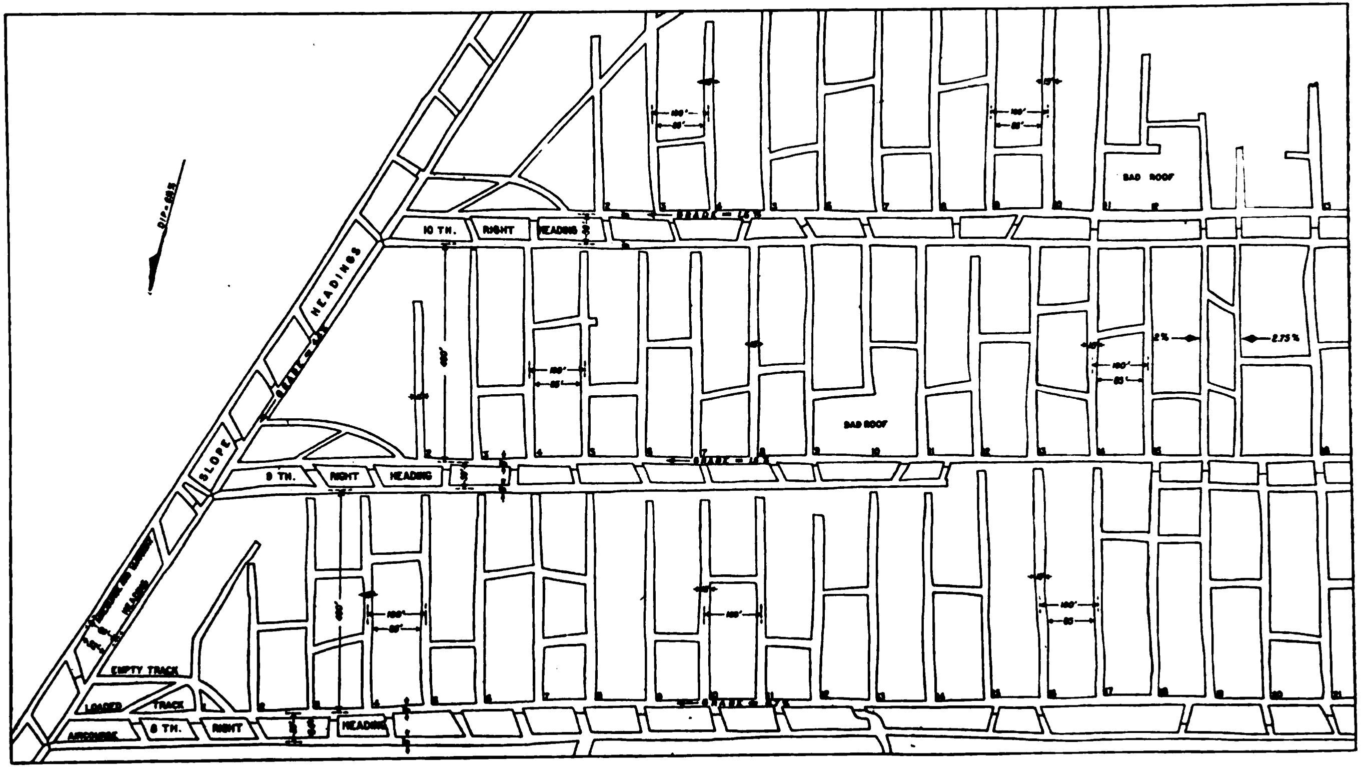

FIG. 29. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1904

these cross-headings. The rooms are driven 15 feet wide on 65-foot centers, leaving pillars 50 feet wide. Twenty-five rooms are driven in each of these diagonal panels. Unusually large protecting pillars are left along the main haulage roads. This system has been found to be especially adapted to rapid gathering of cars thus ensuring a large tonnage. It has been found, however, that a very large recovery from the pillars is impossible, owing to the many sharp angles, which, in a thick seam of soft coal, are always difficult and ofttimes impossible to extract. This sharp-angle method was even resorted to formerly in cross-cutting the pillars preparatory to drawing them, but this has been changed to a rectangular method, thereby increasing the actual percentage of pillar coal recovered from 80 per cent to 83 per cent. The distance of rooms apart has also been increased in the last few years to 100-foot centers giving pillars 85 feet thick. It is expected that the extraction of these will show a further increase in the percentage of yield from pillars. The present yield from headings, rooms, and pillars under this system is about 90 per cent, considering the recovery from headings and rooms as 100 per cent.

Fig. 29 "illustrates a method instituted in the latter part of 1904. The main haulway is an extension of the slope from the opposite side of the basin. Double entries are turned off from this entry, on 1½-per cent grade, 400 feet apart, from which rooms are driven directly on the rise of the coal. Rooms are from 13 feet to 15 feet wide and . . . they are driven at 100-foot intervals, leaving a pillar 85 feet wide. The length of a panel is about 2,500 feet, containing 22 rooms. There are five such panels in this district and when completed it is proposed to draw the pillars in a retreating fashion with the line of pillar work on an angle of 45 degrees across the whole district. A similar method in another district . . . is yielding 88½ per cent from the pillars with a total recovery of 94 per cent from headings, rooms, and pillars . . . the greatest height of the overlying strata is 250 feet."

George S. Brackett states* that in 1898 he made some careful estimates of the percentage of recovery over a period of a year in the Georges Creek region of Maryland. The data for the computations were obtained from two mines which were worked under the following general conditions:

> The thickness of coal was 7 feet, 3 inches; the inclination was 5 to 18 degrees; the system of mining was the room-and-pillar retreating method. All the entries were driven to the boundary before any rooms were opened, and a good line was maintained on the drawing of pillars. No. 1 mine had moderate grades, and a better roof than No. 2. No. 2 had grades

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* Personal Communication.

as steep as 18 per cent, and the roof was decidedly heavy on pillar workings.

The following results were obtained:

	Total Per Cent of Extraction	Per Cent of Pillars Obtained, Including Chain and Barrier
No. 1 Mine	97.6	97.0
No. 2 Mine	82.1	71.3

The average total recovery of the two mines was nearly 90 per cent.

25. *West Virginia.*—The modern methods used in the large mines of West Virginia are among the most advanced found in this country. In many parts of the state, coal mining is carried on by large corporations which are financially able to conduct operations with a view to ultimate economy. In most cases this ability has resulted in the development of methods of operation which lead to very high percentages of extraction.

In the Fairmont district in the northern part of the state, the more recently opened mines are planned for large production and a high percentage of extraction. According to the West Virginia Geological Survey, the Pittsburgh bed, which is the one mined in this district, contains more than 7 feet of clean coal* and the average total thickness of the bed is about 8 feet. The newer mines are projected on the panel system; the last rooms on each room entry are turned first, and the pillars are drawn immediately, a line of break being maintained at an angle of about 45 degrees with the entries. A plan of operation used in this district is illustrated by Fig. 30. The method of attacking pillars is shown in Fig. 31. One of the principal operators in the Fairmont district estimates the recovery, where mines are laid out systematically on the panel system, to be from 85 to 90 per cent of the entire seam.†

A company operating to the south of Fairmont estimates that 85 per cent is a good recovery. In the workings of this company, the rooms represent 27 per cent of the total area; the pillars down to the heading stump, 39 per cent; and the chain and barrier pillars, 34 per cent. The recovery in these three classes of working would be respectively, 100, 90, and 70 per cent. This would give a total yield of 86 per cent. This company has no very accurate figures on recovery.*

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* W. Va. Geol. Sur., Vol. II., p. 180, 1903.

†Smyth, J. G., Personal Communication.

In the mines in the Freeport coal bed in the Piedmont, or Elk Garden, district, there is ordinarily a good shale roof, although there are places in which the shale is replaced by sandstone. In the mines in the Kittanning bed there are from 3 to 14 feet of shale above the coal, and above this are 40 feet of sandstone, although there are places where the sandstone forms the roof for short dis-

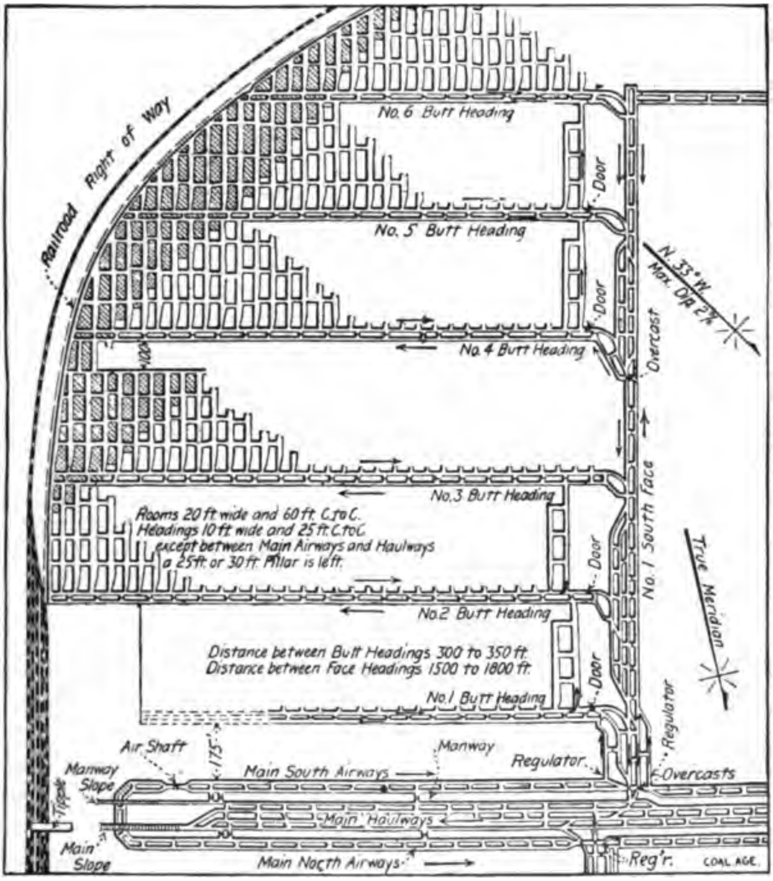

Fig. 30. Plan of Working—Fairmont, West Virginia, District

tances. The percentage of extraction in this region is about 90 per cent; in most of the new mines, however, an extraction of 97 or 98 per cent is being reached. This rate of recovery is higher than it formerly was, because changes have been made in the order of driving rooms and drawing pillars. The earlier custom was to drive long entries, from which rooms were turned on the advance. The room pillars could not be drawn until the entry was finished because of the danger of squeezes, and as this process sometimes occupied four or five years, falls occurred which made it impossible to recover a high percentage of the pillars. Under the present system, room entries are driven long enough for 20 rooms, and the inside room is turned first. Work on room pillars is commenced as soon as rooms 19 and 20 are finished, and room and entry pillars are taken out rapidly. Nearly all the pillar work is done with picks, and there is little machine work carried on in the district. The pillars are attacked by cross-cuts, and a stump about four feet wide

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*Brackett, George S., Personal Communication.

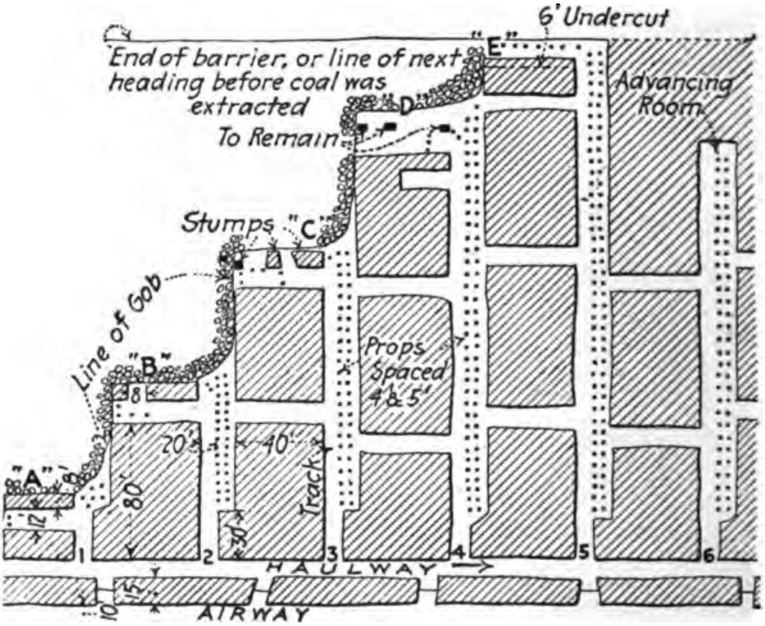

FIG. 31. PILLAR DRAWING IN FAIRMONT, WEST VIRGINIA, DISTRICT

is left next to the end. This stump is removed as soon as the cut through the pillar is finished.*

In the Central West Virginia district the operations are in the Lower Kittanning bed which averages 6½ feet in thickness.† The conditions are almost ideal for a high rate of recovery. The bottom consists of hard shale. The immediate roof is of bone coal 3 to 10 inches thick, above which are shales of varying hardness, 10 feet to 15 feet thick. Above this layer occurs sandstone of an average thickness of 10 feet, and above this, shale and overlying earth. Nowhere is the overburden greater than 90 feet in thickness. The extremely favorable nature of the roof is shown by the fact that a complete break is easily obtained with as few as 3 or 4 rooms. In more recent workings 20-foot rooms are turned on 50-foot centers. The rooms are driven 300 feet long, and a 50-foot pillar is left between the heads of the rooms and the adjoining air-course. This pillar is never pierced except in case of extreme necessity. The 30-foot room pillars are taken out by driving cross-cuts through them every 30 feet retreating. The entry pillars are taken out with the room pillars. If the entry stumps and the barrier pillars at the ends of the rooms are considered, it is estimated that 75 per cent of the coal obtained is taken out as pillar coal and 25 per cent as room coal. The cost of room and of pillar coal is about the same. Bischoff estimated that the recovery is at least 90 per cent, and possibly 95 per cent, although no accurate records have been kept. In view of the unusually favorable conditions, it seems probable that this estimate is correct as there is no apparent reason for loss, except that represented by the small amount of coal which the loaders fail to shovel up. This statement, of course, refers to only the area of actual mining operations.

The Pittsburgh bed is being worked also in Braxton and Gilmer Counties about 75 miles south of the workings just mentioned. While the same method of working is followed, the physical conditions are different, and the extraction is not more than 75 per cent. The coal is 6 to 8 feet thick. The bottom is of fire clay 8 to 15 feet thick, and the immediate roof is of fire clay 2 to 6 feet thick. Above this occurs a sand rock thicker than that found in the neighborhood of Elkins, with a heavier overburden. In this southern district the coast of pillar coal is about three cents greater per ton that the cost of room coal.*

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* Personal Communication.

† Bischoff, J. W., Personal Communication.

The Kanawha region is unlike the fields farther south in that there is a larger number of operating companies with a corresponding lesser concentration of ownership. Because of the number of individual operations it is impossible to give any general or standard method, but the room-and-pillar method is universally used. At least one operator is leaving a large barrier pillar, Fig. 32, and a large room stump for entry protection. The first break-through is driven about 80 feet from the entry, and break-throughs are kept

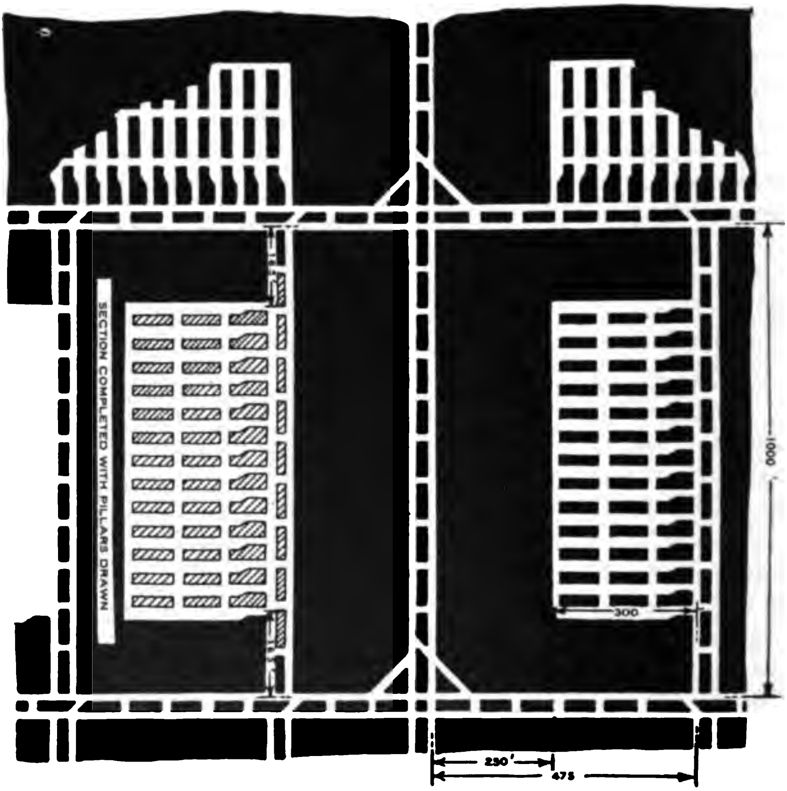

FIG. 32. WIDE BARRIER PILLARS AND ROOM STUMPS, KANAWHA DISTRICT, WEST VIRGINIA

perfectly lined up. Rooms are driven in order, and room pillars are drawn back to the first break-through as soon as the adjoining rooms are completed. The recovery under this method is said to be about 90 per cent and where the roof is extremely good as much as 95 to 97 per cent.*

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* Bischoff, J. W., Personal Communication.

In a paper on the removal of coal from the No. 2 Gas Seam in the Kanawha district, J. J. Marshall reported a very high percentage of recovery and gave the following facts concerning the seam:† The coal bed is made up of two benches separated by a solid parting the thickness of which varies from 10 inches to 40 feet. It has been found that it is not economical to remove this slate when its thickness is more than 24 inches. The aggregate thickness of the two benches averages about 9 feet, the upper bench ranging from 4 feet, 6 inches to 5 feet, 6 inches of clean coal and the lower bench from 3 feet, 6 inches to 4 feet of clean coal. Where it is impossible to mine both benches together, only the upper bench has been taken. The thickness of cover varies from a few feet to 100 feet. After the ordinary method of driving rooms and of drawing pillars on the advance, the mine described has been developed until it is now in position for the butt entries to be worked on the retreating system. On June 30, 1911, the percentage of recovery was said to be as shown by Table 6, nearly all the coal being mined by pick work:

TABLE 6

PERCENTAGE OF EXTRACTION IN KANAWHA DISTRICT

	Total Acres	Percentage of Recovery
High Coal, Both Benches	84.61	91.8
Upper Coal, Upper Bench Only	67.87	98.7
	152.48	94.9

Computations of areas are made from the mine map, and the method of computation does not insure the accuracy of the percentages given. It seems that the values are too high. If there is a loss of 4 feet or more in thickness across the working face, it is recorded, but if the loss is less than this, it is too small to show on the map which is drawn to the scale of 100 feet to the inch, and the recovery is regarded as practically complete. It was said to be seldom necessary to record a loss, especially in the upper coal.

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* Cabell, C. A., Personal Communication.

† Marshall, J. J., "The Removal of Coal from the No. 2 Gas Seam in the Kanawha District," Proc. W. Va. Coal Min. Inst., p. 303, 1911.

In the Cabin Creek portion of the Kanawha district little attention has been paid to the extraction of pillars until recently, and the extraction has amounted to only about 50 per cent. For the past 3 or 4 years, however, the extraction has been about 85 per cent. Since the proper sizes of pillars are now known and the men have a better understanding of pillar work, it is expected that the percentage of extraction will show some further increase.*

Two mines in the New River field, one in the Fire Creek bed and the other in the Sewell bed, have a recovery which is considered practically complete.† At those two particular mines the roof conditions are very favorable; in other sections of the field where they are not so good and where less attention is paid to recovery, it is thought that a fair average extraction is about 90 per cent.‡

The Pocahontas district, in the southern part of West Virginia, is one of the most important coal producing regions in the country, largely because of the high quality of the coal. Pocahontas coal is low in volatile matter and therefore is nearly smokeless; it contains little ash and little sulphur, and it makes an excellent coke. Because of these characteristics there is large demand for it. It is extensively used in coke production, in power plants, in the navy, and in domestic heaters. For coking purposes, however, Pocahontas coal is not used so extensively at present as it was a few years ago. Several beds are being operated, but the principal mines are in the Pocahontas No. 3 bed. The seam varies in thickness from about 4 feet on the west to about 10 feet on the east, but the change is gradual and the thickness is quite uniform within the area of a single mine. This seam has a fire-clay or slate bottom, and a draw-slate roof.¶ It always has one streak of bone about 2 inches thick to which the coal adheres on both sides; consequently when a piece of bone is thrown out, about twice as much coal is lost.

The No. 4 bed, which has two streaks of similar bone, is found 75 to 80 feet above the No. 3. Above this bed occurs a seam of interstratified coal and slate locally termed a "black rash." This rash contains on the average about 25 per cent of ash, and it is considered worthless, but sometimes over rather large areas there occurs in it a streak of clean coal from 6 to 8 inches in thickness. Miners are supposed to leave all this rash in their working places, and most of it is left, but sometimes some of this clean coal is loaded out. This fact will explain the higher yield from the No. 4 bed. In other words, the coal actually mined and loaded sometimes has a thickness greater than that considered in calculating the contents of the bed. The fact that a considerable amount of good coal is lost with the bone explains the smaller yield in mines in the No. 3 bed. It is believed that if it were not for this loss the extraction would average about 95 per cent.

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* Keely, Josiah, Personal Communication.

† Personal Communication.

‡ Cunningham, J. S., Personal Communication.

¶ Eavenson, H. N., Personal Communication.

As a rule, throughout southern West Virginia, the coal lands are held by land-holding companies, which lease to operating companies. The royalty is generally 10 cents per ton of coal and 15 cents per ton of coke, with a yearly minimum.

The beds are nearly flat and quite regular, of an almost ideal mining height, generally with good roof and bottom, and with little gas and water in the drift mines. It is possible, therefore, to lay out a definite plan of mining in advance, and to follow such a plan more closely than in sections where natural conditions are less favorable. In many cases the landholders specify that the coal shall be mined in accordance with certain plans, and prescribe a minimum of extraction. Certain departures from the standard methods, however, are permitted where it seems advisable.

Fig. 33 illustrates the plan of development formulated by the Pocahontas Coal and Coke Company,* which may be carried out by one of three possible procedures as follows:

Panel No. 1.—Drive rooms on 3rd cross entry as soon as come to, begin robbing as soon as second room is completed and rob advancing on 2nd and 3rd cross entries to within 100 feet of 2nd cross entry, on 1st cross entry drive last room first and rob retreating as shown, taking out the barrier pillar left on 2nd cross entry.

Panel No. 2.—Drive entries to the limit before turning rooms except as shown, turn last room on 3rd cross entry first, begin robbing at inside corner of panel, develop rooms only fast enough to keep in advance of robbing and bring robbing back with uniform breakline until completed to barrier pillars.

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* Stoek, H. H., "Pocahontas Region Mining Methods," Mines and Minerals, Vol. 29, p. 395. Stow, Audley H., "Mining in the Pocahontas Field," Coal Age, Vol. 3, p. 594, 1913.

Panel No. 3.—Continuous panel, drive entries to the limit before turning rooms except as shown, turn last room on 1st cross entry first, begin robbing as soon as second room is completed, develop rooms only fast enough to keep in advance of robbing, and bring robbing back with uniform breakline until limit of mining is reached.

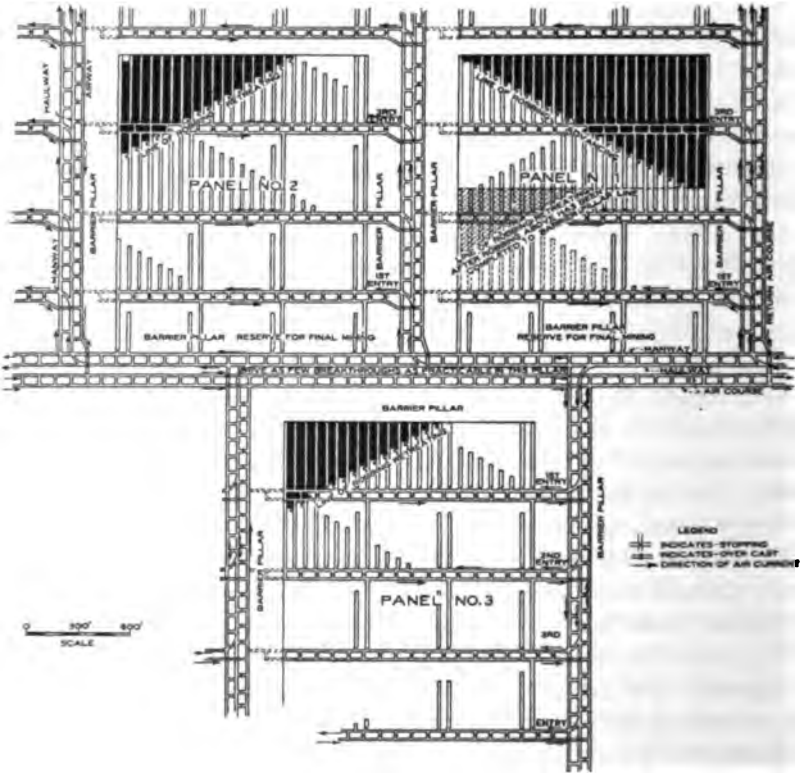

FIG. 33. PLAN OF WORKING OF POCAHONTAS COAL AND COKE COMPANY

According to W. H. Grady, chief mine inspector of the Pocahontas Coal and Coke Company,* the essential advantages of this plan of mining include: provision for tonnage during the development period, provision for meeting the market demand, large barrier pillars insuring against squeezes and rendering impossible the destruction of coal over an extended area, 4-entry system for all extensive main entries with two as intakes and two as returns with breakthroughs intervening only at the points where the cross entries turn off, rendering unnecessary the building of expensive masonry brattices every 80 feet, and insuring the maximum quantity of air for ventilation at a minimum cost for brattices and ventilating power, and cross entries with narrow chain pillars which permit the rapid advance of the entry.

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* Grady, W. H., "Some Details of Mining Methods with Special Reference to the Maximum of Recovery," W. Va. Coal Min. Inst., Dec. 1913; Coal Age, Vol. 5, p. 156, 1913.

The success of this method, with regard to high output, is shown by the values given in Table 7, taken from Grady's article. The percentage of recovery is based on the thickness of the total seam, including the portion rejected. The lower percentages of extraction shown in Table 7 were reached where pillars were being robbed after standing for many years. Operations had not proceeded far enough at the date of the paper quoted to permit a definite statement as to how great the final recovery would be. A later statement* has been received to the effect that the mines of the lessees of the Pocahontas Coal and Coke Company will probably show an average recovery of 90 per cent. Since some of these operations have extended over many years and since they were not so well managed formerly as at present, it is probable that the recovery now is more than 90 per cent.

In the Pocahontas district, much attention has been given to the subject of recovery, and the values which have been supplied by operating companies are as accurate as such values can be made. H.

TABLE 7

RECOVERY OF COAL IN MINES OF POCAHONTAS COAL AND COKE COMPANY

Plant	Thickness of Seam in Feet	Acres of Entry Mined	Acres of Rooms Mined	Acres of Pillars Mined	Total Acres Mined	Total Tonnage Mined	Tons Mined per Acre	Theoretical Tons per Acre	Percentage of Recovery	Proportion of Seam Rejected
1	6.15	3.06	4.57	11.03	18.66	165,254	8,856	9,922	89.3	0.24
2	5.65	4.40	4.80	14.80	24.00	188,391	8,185	9,115	89.79	0.22
3	5.16	2.68	6.52	15.80	25.00	180,386	7,215	8,325	86.6	0.22
4	4.42	5.88	8.65	13.09	27.62	192,437	6,960	7,131	97.6	0.23
5	5.94	7.00	10.09	19.20	36.29	334,005	9,203	9,582	96.0	0.22
6	4.32	2.11	3.64	9.20	15.04	94,427	6,278	6,969	90.0	0.31
7	5.34	3.31	6.34	0.00	9.65	83,000	8,601	8,614	99.8	0.20
8	5.42	3.72	6.06	9.72	19.50	144,769	8,181	8,777	93.2	0.20
9	4.65	8.10	16.80	2.34	27.24	201,044	7,380	7,534	98.0	0.18
10	8.03	5.20	8.47	10.09	23.76	262,975	11,068	12,923	85.6	0.23

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* Eavenson, Howard N., Personal Communication.

N. Eavenson, whose communication has already been referred to with regard to the character of the beds mined, says that the measurements of areas worked out are as close as it is possible to get them on a large scale. The thicknesses given are those of the clean coal, and do not include any bone or black rash. In many instances a record has been kept of coal left in small areas, and the values shown by these tests agree very closely with those given in Table 8. It will be seen from the table that the amount of extraction for mines 9, 10, and 11 is given as more than 100 per cent. This record is explained by the statement previously made concerning the loading out of coal supposed to be left in the mines. At No. 9, the rash is much cleaner than at the other mines of the company, and while the seam is thicker, it carries only a very small amount of dirt; thus a higher percentage of clean coal is given.

TABLE 8

STATEMENT OF THICKNESSES AND RECOVERIES, ALL MINES, UNITED STATES COAL AND COKE COMPANY 1902 TO 1916, INCLUSIVE

Mine No.	Area Worked Out per Cent		Average Thickness Clean Coal	Net Tons Recovered Per Acre Foot	Percentage of Recovery	No. of Seam Worked	Date of First Shipments
	Rooms and Entries	Pillars					
1	45.5	54.5	5.52	1746	97.0	3 & 4	1903
2	59.7	40.3	5.67	1790	99.4	4	1902
3	61.2	38.8	4.56	1429	79.4	4	1903
4	57.3	42.6	6.19	1581	87.8	3	1904
5	63.5	36.5	6.88	1606	89.2	3	1904
6	63.6	36.4	6.09	1769	98.3	4	1903
7	66.9	33.1	6.27	1770	98.3	4	1905
8	70.8	29.2	5.98	1728	96.0	4	1905
9	74.2	25.8	7.24	1908	106.0	4	1906
10	85.1	14.9	5.31	1806	100.3	3 & 4	1907
11	73.3	26.7	5.26	1807	100.4	3 & 4	1907
12	69.8	30.2	8.22	1622	90.0	3	1906
	65.9	34.1	5.95	1738	96.5		

At the No. 3 mine, which shows the lowest percentage of extraction, the roof is exceedingly bad. Above the coal there occurs a layer of shale and slate, from 5 to 10 feet thick, which it is impossible to support even by close timbering. The mining practice is fully as good at this mine as at the others, but the yield is much less because of the more difficult conditions. The table shows the areas worked out at different mines and the percentages recovered. It is Eavenson's opinion that the average recovery in the larger mines throughout the Pocahontas field is fully 90 per cent, and that in many mines even a higher figure is reached.

One of the landholding associations of the Pocahontas district furnishes information* concerning some of the operations of its lessees, and submits a table (Table 9) showing the percentage of recovery. It takes account of operations up to January 1, 1917.

TABLE 9

PERCENTAGE OF RECOVERY ON LIVE WORK AND ROBBING
POCAHONTAS DISTRICT

January 1, 1917

AREA MINED			PERCENTAGE OF RECOVERY			
No. of Mine	Live Work, Per cent	Robbing, Per cent	Live Work (Assumed)	Gross Robbing	Net Robbing, Assuming that 37½ per cent of Gross Robbing was Mined as Live Work	Total to Date
1	16.1	83.9	97	84.6	77.2[1]	86.6
2	.3	99.7	97	73.4	59.2	73.5
3	9.4	90.6	97	81.2	71.7	82.7
4	24.2	75.8	97	80.6	70.7	84.6
5	38.3	61.7	97	76.9	64.8	84.6
6	12.1	87.9	97	87.0	81.0	88.2
7	15.5	84.5	97	77.5	65.8	80.6
8	15.4	84.6	97	79.3	68.7	82.0
9	51.8	48.2	97	83.3	73.9	90.4
10	53.7	46.3	97	83.4	75.2	90.7
11	63.5	36.5	97	86.0	79.4	93.0
12	51.7	48.3	97	81.9	72.8	89.7

[1] Values obtained by assuming that 37.5 per cent of the pillar work is done under the same conditions as live work, i.e., with recovery of 97 per cent; thus $0.625 X + 0.375 \times 97 = 84.6$, $X = 77.2$.

It will be noted that the extraction at mine No. 2, the first of the leases to exhaust the No. 3 Pocahontas seam, is not more than 73 or 74 per cent. In addition to the losses which will be mentioned, there was a considerable loss here of top coal. A thickness of 18 to 24 inches was left up in the first mining with the expectation that it would be recovered on the retreat, but most of this coal was ultimately lost on account of the bad roof. It is also possible that the loss in the coke yard at this plant, where the maximum number of ovens was run in proportion to the output, amounted to almost double the average tabulated amount.

An inspection of the table shows that, in most cases, the highest percentage of recovery has been reached at those mines where the pillar work has been least in proportion to the live work, that is, at those still in the earlier stages of working. Future operations in these mines may be expected to lower these values, but it would seem that 80 per cent would be a very conservative estimate of the average amount of coal that should be won in the various mines of the property up to the exhaustion of all properties under lease.

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* Lincoln, J. J., Personal Communication.

J. J. Lincoln has discussed losses in the Pocahontas field, and the facts brought out are of general interest in connection with the subject of coal recovery. It is said that losses may be considered under three headings; (a) mixing of coal with refuse, (b) loss in drawing pillars, and (c) loss in coke making.

There is loss in removing the bone and pyrite from the coal, as some coal adheres to the refuse. Under the present mining methods, this loss is from 2 to 4 per cent, and occurs in both new work and robbing.

The following losses are to be expected in pillar drawing in addition to loss of coal attached to refuse: (1) In drawing each stumps are occasionally crushed by the pressure of the top before broken rock from the adjacent gob from covering the coal. This loss will run from 3 to 10 per cent, according to conditions. (2) When the pillars are drawn by splitting, a similar loss, frequently greater, occurs. (3) As the drawing progresses small sections of stumps are occasionally crushed by the pressure of the top before they can be removed. (4) Stumps, sections of pillars, or entire pillars may be crushed by the weight before they can be removed, or may be surrounded and cut off by the broken top. In the mines where actual losses from this source are closely recorded they do not reach 1 per cent, and there is no mine in which they will reach 2 per cent. The third loss is not directly chargeable to mining, but occurs in the making of coke. Where the tonnage of coke produced is used as a measure of the amount of coal taken out, this loss becomes signifieant in calculating the percentage of coal won. The ratio used by the company in all calculations of tonnage has always been 1.6 tons of coal to one ton of coke. This ratio assumes an actual average yield of 62½ per cent of coke, but in practice this yield is not obtained, the average yield under existing conditions being nearer 55 per cent. This loss has always been charged, with the other losses, directly against the mining. This ratio cannot be used directly in determining the actual amount of coal mined, because only a part of the coal is coked. The loss varies in amount from 1 to 5 per cent, according to the conditions in the different operations, and the actual percentage of extraction is slightly higher than that given in the table because of this error.

Losses in mining and coke making may then be tabulated as follows:

"(a) Removing Refuse—Coal thrown into gob with bone and sulphur bands	2 per cent to 4 per cent
(b) Robbing:—	
1. Stumping, or Splitting pillars	2 per cent to 10 per cent
2. Sections or Stumps	0 per cent to 2 per cent
3. Pillars lost	0 per cent to 2 per cent
(c) Additional coal consumed in coke making over and above the amount covered by the constant of calculation, 1.6 tons equals 1 ton coke	1 per cent to 5 per cent
Total	5 per cent to 23 per cent"

A peculiar system followed by the Gay Coal and Coke Company of Logan, West Virginia, and called a single-room system[*] has resulted from an attempt to apply the long-wall method to a seam the average thickness of which is 5 feet, 7 inches. This seam dips to the southwest about 1½ per cent, is practically free from partings, and is of the nature of splint coal, the bottom bench being rather strong, and the top bench somewhat friable. The average thickness of cover does not exceed 500 feet while the maximum is less than 1,000.

A block of coal was cut by two entries 600 feet long, Nos. 4 and 5 (Fig. 34). These were connected at their extremities, which where 300 feet apart. The purpose was to take out the block of coal thus formed in a retreating direction by commencing at the inner end and by working outward with a face about 300 feet long. One hundred beech or hickory posts were used to support the roof near the face. The top of each was covered with 1-inch poplar, and the bottom with 1/16-inch sheet steel. Each post was mounted on a hydraulic head weighing about 700 pounds and tested to a pressure of 3,000 pounds per square inch. These were set along the face 6 feet apart in parallel rows 3 feet apart. The cost of the equipment was approximately $5,000.

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[*] Gay, H. S., "A Single-room System," Proc. Coal Min. Inst. Amer., p. 157, 1906; Mines and Minerals, Vol. 27, p. 325, 1906; Personal Communication.

When the walls became more than 30 feet apart, rows of props on 15-foot centers were set 8 to 10 feet apart. When the distance between the walls had reached 60 feet, the portable posts were put into use, a row of 50 posts on 6-foot centers being set 10 feet from the face. The heads were covered with wooden cap pieces, and the plungers were raised by a pressure of 50 pounds per square inch. When the face had advanced 6 feet farther, the other 50 posts were put in; as the work progressed, the first row was moved 6 feet ahead of the second, the posts being moved one at a time. An occasional row of posts similar to the first was also set as a precautionary measure.

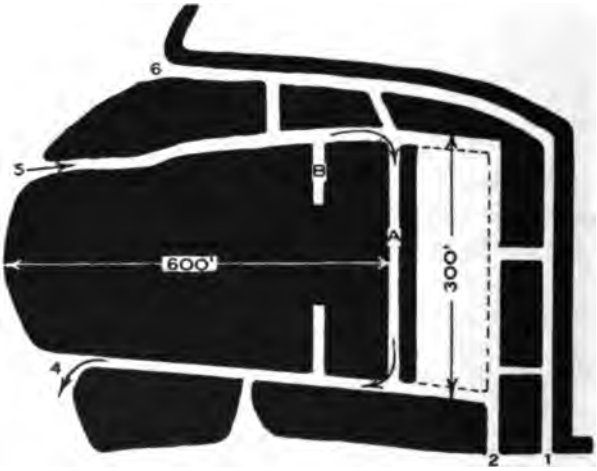

FIG. 34. SINGLE ROOM METHOD, LOGAN COUNTY, WEST VIRGINIA

When the walls were 100 feet apart, it was thought advisable to blast down the roof. The portable posts were set in a single row 6 feet from the face. Examination after the rock had fallen showed that the immediate roof consisted of a seam of strong sand slate at least 30 feet thick without any sign of a parting and that difficulty would be encountered in attempting to apply the long-wall method in this mine.

The advantages to be derived by working on a long face were so great that the company devised another plan which involved working out the remainder of the block by a system of rooms 80 feet wide, parallel with the long-wall face and separated by 30-foot pillars. Each room was to be opened by driving a sub-entry across from entry No. 4 to entry No. 5 (Fig. 34), and thereafter the manner of working was to be identical in every respect with that of the long-wall system.

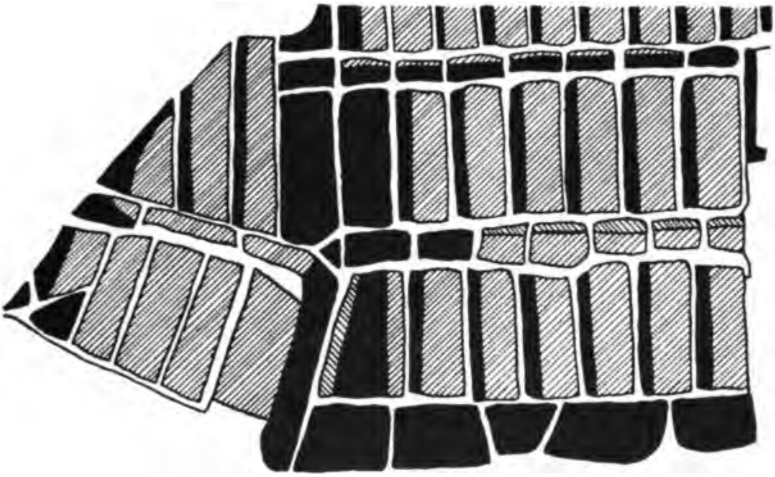

FIG. 35. BIG ROOM METHOD, LOGAN COUNTY, WEST VIRGINIA

From the experience gained it was thought that, with the roof in normal condition, rooms could be worked 90 feet wide with 30-foot pillars. Entries Nos. 4 and 5 were therefore continued eastward, and sub-entries were spaced for the rooms as shown on the map (Fig. 35). The last of the sub-entries was widened to 40 feet, and a single row of ordinary props, 8 to 10 inches in diameter, was set close to the face on 15-foot centers. When the face had moved 10 feet farther, a second row was set. When the room had reached a width of sixty feet, a row of portable posts was set on 10-foot centers, and at 70 feet another row was set. The ordinary props were used for detecting the action of the roof.

One of the advantages of this system of mining is that a large amount of coal per employee may be obtained. In fact the production per man is considerably greater than with the room-and-pillar method, and greater even than would be possible with the long-wall method. The highest rating in this seam for car distribution, exclusive of this mine, is 13 tons per loader; this mine is rated at 20 tons per loader. It is the opinion of Mr. Gay that it would be possible to produce about 11 tons per inside employee per 9-hour day for five days a week. The method also results in the recovery of a very high percentage of coal. A calculation based upon the number of tons shipped and the area excavated, according to planimeter measurement, indicates that the extraction was 85.9 per cent.

Since this description was published, the system has been modified to reduce the narrow work, but the general plan has been followed. Instead of driving a single room to form the working face, parallel rooms separated by an 18-foot pillar are driven; thus the use of brattices is unnecessary, and ventilation is improved. The hydraulic posts were soon abandoned, as posts without the hydraulic heads are cheaper, and they are easily recovered. One of the most important facts concerning the operation is that there has not been a single fatal accident in the mine since work was begun.

Another system in which an effort was made to obtain the advantages of long-wall working was tried a few years ago in West Virginia.[*] In developing this system (Fig. 36) triple entries are driven from the outcrop, near which double entries are turned off at right angles. From these, entries are driven parallel with the main entry; thus blocks of coal about 900 feet wide are cut off. Block entries are turned from the main entry and from these side entries, parallel with the cross entry, spaced about 500 feet apart, and driven for about 800 feet; thus the coal is blocked into areas approximately 500 by 800 feet. In working these blocks, a room is turned first at the end of the block entries to form a working face for the long-wall machine. The blocks are then worked back toward the main entry for 500 feet; thus a barrier of 300 feet protects each main entry. Track is laid along the face as near the coal as possible, and is moved as the face progresses. The roof is allowed to fall, but the line of break is kept at the correct distance from the face by three or more rows of props, the last row being moved forward after a cut is made. The trolley wires are hung on hangers as usual, but to keep them tight they are carried on portable drums. Current for the motor and machines along the face is taken from the trolley wire on the entry by means of a cable which is also coiled on a portable drum. Ventilation is controlled by placing a regulator in the return air course of each block. Each block or face is provided with a separate supply of fresh air by having overcasts placed at the air courses to admit the return air into the main air course entry. No doors are required at any points in the mine.

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[*] James, W. E., "Block System of Retreating Long-wall," Proc. W. Va. Coal Min. Inst., p. 187, 1911; Cabell, C. A., Personal Communication, and Patent Specifications.

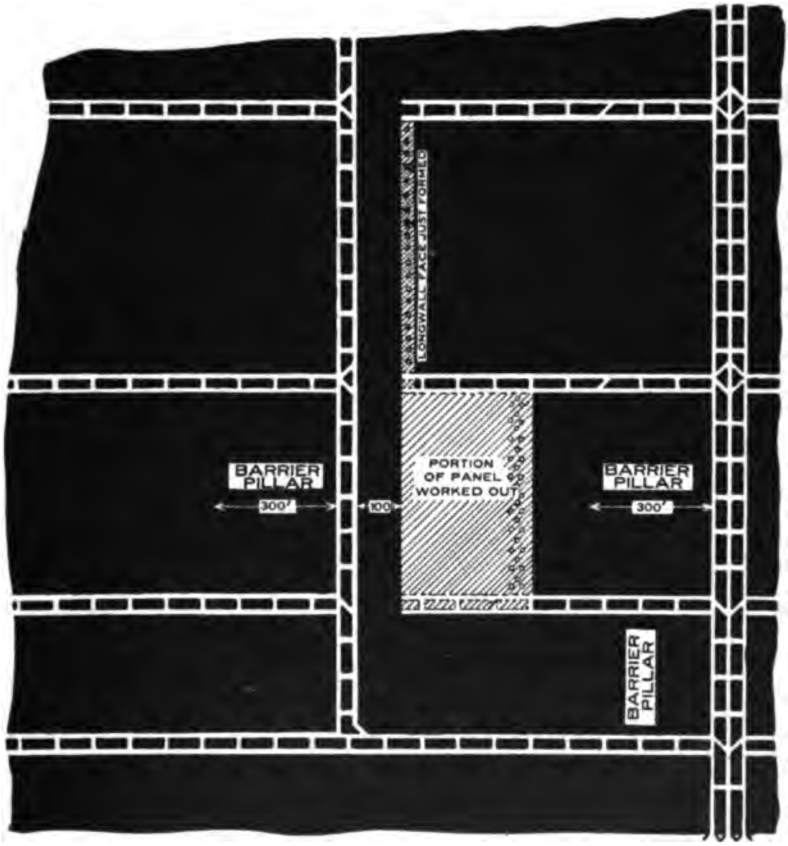

FIG. 36. BLOCK SYSTEM OF RETREATING LONG-WALL, WEST VIRGINIA

It was claimed for this method that a block will produce 440 tons per day with the use of only 500 feet of track in addition to that on the entries, that practically all the coal is obtained, and that the work of the rolling stock and cutting machines is concentrated. The trial of this method was temporarily abandoned because of an inadequate car supply, but the work was considered successful. It was not carried far enough to provide reliable data for an estimation of total extraction.

26. *Ohio.*—The literature of coal mining contains little information regarding conditions in Ohio. Some of the operations in the Hocking Valley district have been described in articles which state that large quantities of coal are left in the roof because of the poor quality of the product. An article on Hisylvania Mine No. 23 states* that the bed mined in the Hocking Valley district is the Middle Kittanning, Hocking Valley, or No. 6. The bottom consists of a few inches of fire clay overlying hard rock. The roof is of shale, 6 to 8 feet thick. The coal bed consists of three benches. The thickness of good coal is about 6 feet, and above this is about 5½ feet of a poorer coal separated from the lower portion of the bed by a distinct parting. This upper bed, with the upper bench of the lower bed, is known as top coal. In this district all coal in excess of 6 feet, and in many places in excess of 4½ feet, is to be credited to this upper bench which has a maximum thickness of 10 feet.

James Pritchard† estimates the percentages of extraction in the districts as follows:

In the Pittsburgh vein district, the Cambridge field, and the Hocking field, districts which produce approximately three-fourths of the coal of the state, the rate of extraction will range from 60 to 70 per cent. In the Massillon and Jackson fields, the rate of extraction may reach 85 per cent. In the Deerfield and Mahoning districts, the rate of extraction may reach 85 per cent. Throughout the remainder of the state, the maximum percentage of extraction will run from 60 to 70 per cent. The average rate of recovery is approximately 60 per cent, with a minimum of 55 per cent and a maximum of 75 per cent.

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* Burroughs, W. G., "Hisylvania Mine No. 23," Coll. Eng., Vol. 34, p. 421.

† Pritchard, James, Chief Deputy and Safety Commissioner of Mines, Personal Communication.

The average value represents the most common percentage of extraction.

The conditions in the Pittsburgh vein district of eastern Ohio are described by Roby* as follows:

The extraction is limited by various physical and commercial conditions. The roof consists of a bed of weak coal of poor quality, variable in thickness, and separated from the main bed by a layer of drawslate which disintegrates on exposure. Overlying the roof coal is an unstratified "soapstone" 4 to 8 feet thick, and above this is a thick layer of hard limestone. The country is hilly, and the total cover varies from 30 to 600 feet in thickness. The character of the roof makes it necessary to leave larger pillars than would be required under a good roof. The roof coal is left up, and as it is poor in quality and not marketable, it is not considered in making estimates of extraction. The room-and-pillar method is used. There has been much discussion concerning the possibility of applying the long-wall method, but the fragility of the roof and the tendency of all rock below the limestone to shear off at the solid face have seemed to make the method impracticable.

It is desirable that rooms be worked out as quickly as possible because of the tendency of the roof and pillars to fail. Because of these conditions, the numerous interruptions which have been caused by strikes and business depressions have tended to make the rate of recovery lower than would have been the case with uninterrupted operation.†

J. C. Haring states that the recovery in the Massillon district has not exceeded 75 per cent. The highest recovery in the district is probably at the Pocock No. 4 mine where it is estimated that fully 90 per cent of the coal has been obtained.

At Steubenville, the mine of the LaBelle Iron Works has been operated on the long-wall system since 1913; prior to that time the room-and-pillar method was employed.‡ The bed is the Lower Freeport which is a little over 3 feet in thickness and has a good shale roof.

At present a considerable amount of stripping is being done in the No. 8 coal in the vicinity of Steubenville. The coal outcrops on the slopes of hills, and the rapid increase in the thickness of the overburden restricts the operations to a narrow belt, although they may extend for a considerable distance along the outcrop.

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* Roby, J. J., Personal Communication.

† Haring, J. C., Personal Communication.

‡ Burroughs, W. G., "Long-wall Mining at Steubenville, Ohio," Coal Age, Vol. 11, p. 697.

27. *Kentucky.*—The development of coal mining in Kentucky has been comparatively recent, and some of the Kentucky fields are still so young that it is impossible to obtain estimates of recovery. The state may be divided broadly into two districts, the western district being closely allied with the fields of Indiana and Illinois, and the eastern district with those of West Virginia and Virginia.

The statements presented in the following paragraphs concerning the western district are made on the authority of N. G. Alford,* who says that the fields contain about 38.3 per cent of the coal-bearing areas of this state and that in 1912 47.7 per cent of the total production came from this district. The smallness of many operations is shown by the statement that 21 per cent of the mines produced less than 10,000 tons per annum each; 51 per cent produced less than 60,000 tons each; 23 per cent produced more than 100,000 tons each; and two companies, operating 18 mines, produced 2,750,000 tons each.

Generally, the rate of recovery in the mines of western Kentucky is about 66 2/3 per cent, although in some instances it is as low as 44 per cent. Without an exception the mines of this district are developed on the room-and-pillar system with double or triple entries. With the exception of two or three isolated operations, all the coal is produced from three seams.

Most of the coal comes from the No. 9 and No. 11 beds, the former producing about three-fourths of the total output of the field. This bed is present in eight counties and approaches 5 feet in thickness. In most places it is reached by shafts of 300 feet or less in depth, although there are some local surface depressions which permit access by slopes or drifts. It has a black shale roof and a soft fire-clay bottom.

The No. 11 seam lies from 40 to 100 feet above the No. 9, and follows the latter in commercial importance. Its average thickness is 6 feet. Above the coal is a stratum of limestone of thickness varying from a few inches to 40 feet. This limestone is usually separated from the coal by a thin stratum of heavy laminated clay 6 to 24 inches thick. This top adheres uncertainly to the limestone above it, and presents a constant danger. Near the outcrop the top becomes very treacherous. The bottom of this seam consists of soft fire clay which frequently heaves in haulage entries that have been opened for some time. About half the mines in this seam are shaft mines, and the remainder are drift mines.

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* Alford, Newell G., "Problems Encountered in Kentucky Coal Mining," Ky. Min. Inst., 1913; Coal Age, Vol. 5, p. 674, and Personal Communication.

The third seam in commercial importance is No. 12, which is found best developed in Clay and Webster Counties. Its approximate depth below the surface is 225 feet. Its average thickness is 7 feet. The bottom is of fire clay which is high in calcium and which disintegrates rapidly when drainage water is directed through it in ditches. The roof consists of light gray disintegrated shale 10 to 15 feet thick. If all the coal is removed, this top will fall to a height of 6 or 8 feet, and heavy timber sets, thoroughly and solidly lagged, are required to support it. Because of this condition it has been found necessary to leave 16 inches of top coal as a roof. Sixty per cent of this top coal is recovered from rooms, but no attempt is made to recover it from entries. When the development of the No. 12 seam was begun, rooms were driven 21 feet wide on 33-foot centers; but this width of pillar was found to be too narrow, and it has been increased to 20 feet, with rooms 21 feet wide. Under these conditions a recovery of 44 per cent is the best which has been reached up to this time. This low percentage of recovery is in part due to physical conditions, and in part to over-development and keen competition.

Several factors contribute to limiting the recovery in the No. 9 and No. 11 beds. The most important of these is probably inadequate planning of future workings. Frequently pillars are left too small; consequently the bottom heaves, and the pillars are crushed. Partial recovery of pillar coal by taking slabs off the ribs is not general, and the total recovery of pillars has not been attempted. The operators in this district hold the opinion that pillar robbing in the No. 11 seam is particularly hazardous and impractical, because heavy limestone overlies the seam; in the Connellsville district of Pennsylvania, however, pillars are successfully drawn under a heavy limestone. Alford expresses the opinion that if the workings in the No. 11 seam were properly laid out and started, little difficulty would be found in increasing the percentage of recovery.

In this district there is an over-production with a limited market; consequently competition is keen. When business conditions are normal, the margin of profit is so small as to preclude any costly improvements, and little money has been expended in experiments. Another source of loss lies in the waste at the tipples in the summer, because the demands of consumers are more exacting in times of dull markets. Good coal attached to lumps of pyrite is often discarded in large quantities, and so far this waste has been accepted as unavoidable. This district furnishes one of the best illustrations of the effects of over-development and lack of harmony of interests. There are, however, some operations which are carried out on a considerable scale and with careful attention to the proper planning of the work.

The estimates covering production mentioned previously are confirmed by a personal communication from another operator, S. S. Lanier, who has been a close observer in the district for thirty years and who estimates that the extraction is about 65 per cent.

The eastern Kentucky district is of such recent development that estimates of production are not very reliable. H. D. Easton, operating in the southeastern part of the state, thinks it safe to say that a recovery of 90 per cent is being reached in the Straight Creek seam in Bell County, but much trouble has been experienced from squeezes due to lack of systematic working.

Mines are operated on the room-and-pillar system with rooms turned from both sides of the room entries. Rooms are generally 40 feet wide with 20-foot pillars. Cross entries are driven about 1,200 feet apart, and these usually extend to the property line or to the outcrop. It has been the practice to extract the pillar coal on the retreat, and so far as possible to keep the face lined up over a sufficient distance to get a fall of roof. Room tracks are swung across the face of the pillar and are moved as the pillar is drawn back. If the pillars are narrow, the room tracks are not moved even though the coal has to be shoveled 15 or 20 feet.

There has been no very systematic work in pillar recovery in the southeastern part of the state, and pillars or stumps have been left scattered promiscuously, with the result that many costly squeezes have occurred. One company has lost an entire mine as a result of this practice. It has been the general opinion that it would be impossible to get a clean break in the overlying strata because of the solid sandstone above the coal, but such breaks have been obtained very successfully, even in rather limited areas.

The Harland district, the Hazard district, and the Elkhorn district are all too newly developed to provide a reliable basis for estimating recovery, but it is possible that the percentage of recovery will be very high.

In eastern Kentucky the surface is of no great value, and it is almost invariably owned by the coal companies so that the necessity for sustaining it is not a factor affecting the percentage of extraction.

There are some operations in the central southern part of the state, but the mines are not yet sufficiently developed to yield adequate data for reliable estimates of extraction.* The mines are opened by drifts, and as the coal is irregular, the hills have been entered at many points. A heavy sandstone occurs between the two seams worked, and pillars have not been drawn in the lower seam, because it has been feared that the upper seam would be damaged. In the upper seam, pillar drawing has not been practiced to any great extent, because when tried, it has resulted in breaks extending to the surface through which considerable water has entered. It is the intention of the operators to extract the pillars when the mines have been worked out, and the final percentage of extraction will probably be high.

28. *Tennessee.*—Little information is available on the percentage of recovery in Tennessee, and the statements obtained are not altogether in agreement. One operator,† formerly connected with the industry in Tennessee, states that a few years ago the mining practices were not good. On the first mining, about 50 per cent of the coal was taken, and the ultimate recovery was probably about 80 per cent. Because of the low value of coal lands, less effort is made to get a maximum recovery than in some other districts where coal lands are more valuable.

R. A. Shiflett,‡ Chief Mine Inspector, says that it would be difficult to give any general percentage for extraction since the coal measures vary in dip from horizontal to 40 degrees, and in some cases from horitzontal to vertical. In a large number of mines it is impossible to map out any definite method of mining, and conditions have to be met as they are encountered.

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* Butler, J. E., Personal Communication.

† Coxe, E. H., Personal Communication.

‡ Personal Communication.

Nearly all the drift mines are developed on the double-entry room-and-pillar system. Rooms are driven from 250 to 300 feet, and room pillars are drawn as soon as the rooms are finished. If conditions are favorable, the entry pillars and room stumps are recovered on the retreat; and where the coal is practically level, 40 to 50 inches thick and with good roof and bottom, about 90 per cent is extracted under careful management. This percentage is not reached in many mines because of lack of attention to high extraction. It is thought that the extraction in general does not exceed 65 per cent, but that this percentage could be greatly increased by proper methods and careful management.

One company,* whose method was to turn rooms on the advance and immediately to draw room pillars back to within 65 or 70 feet of the entry on completion of the room, obtained a recovery of nearly 90 per cent up to about the beginning of 1916. At that time four cross entries were lost from heaving of the soft bottom. The cover is 500 to 800 feet in thickness, the coal is 56 inches in thickness, and the bottom is of soft fire clay from 4 to 7 feet in thickness. This company is planning the introduction of the long-wall method. A face of about 300 feet will be formed by connecting the ends of two entries. It is thought that the single stick timbering, with perhaps an occasional crib, will be sufficient. It is expected that the bottom will heave and reach the roof as the latter bends down. The scarcity of labor and the irregularity of the car supply make the success of long-wall operations somewhat doubtful; and if it is necessary temporarily to abandon this method, another which is illustrated in Fig. 37 will be adopted. In this method apparently a little more than 50 per cent of the coal would be taken out from rooms, and the ultimate percentage of extraction should be almost complete. The method will permit concentrated working, and much of the trouble due to the conditions of the floor and roof will probably be avoided.

29. *Alabama.*—Although Alabama† is an important producer

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* Hutcheson, W. C., Personal Communication.

† Strong, J. E., "Alabama Mining Methods," Mines and Minerals, Vol. 21, p. 195, and Personal Communication.

of coal, the working conditions are not so good as those of Pennsylvania, Kentucky, and some other states. One of the distinctive features of the Alabama coal field is that although there are five seams of coal, rarely more than one of them is workable at any one place; in one portion of Jefferson County, for instance, the Pratt seam, which is considered the topmost workable seam of the Alabama coal measures, may have a working thickness of 4 feet, while at a distance of two or three miles the same seam may not be more than 2 feet thick. The coal beds, including the Pratt bed, vary rather abruptly within a few miles with regard to thickness, impurities, and character of roof.

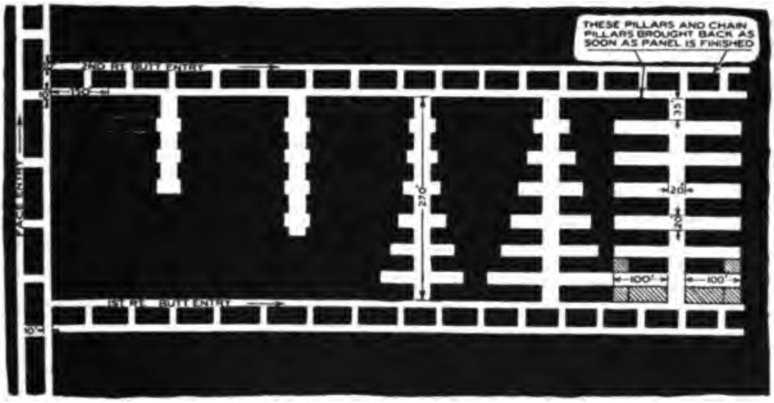
THESE PILLARS AND CHAIN PILLARS BROUGHT BACK AS SOON AS PANEL IS FINISHED
2ND RT. BUTT ENTRY
1ST RT. BUTT ENTRY
ENTRY
270'
35'
20'
20'
100'
100'
10'

FIG. 37. PROPOSED PLAN OF WIND ROCK COAL COMPANY, TENNESSEE

The Pratt seam has been worked longer and more extensively than any other seam in this district. It is probable that the recovery from this seam, under the best methods of working with the room-and-pillar system, is about 87 per cent. This percentage applies only where the thickness is three feet or more; coal thinner than this cannot profitably be removed.

The Mary Lee seam, which is supposed to contain the thickest workable coal, lies about 300 feet below the Pratt seam, and is the one that the operators of the Birmingham district expect to work during the next twenty years. So far as it is known, the thickness of the bed ranges from 6 to 10 feet. It is difficult to get reliable estimates of the recovery of coal in this seam, but one operator reports, on the basis of an experience of twenty years, that the recovery has been about 90 per cent.

There are mines on lower seams in other localities, but little attention has been paid to the extraction of a high percentage of the coal. The operation of these mines depends largely upon market conditions, and they are probably not operated, on the average, more than half the time. Under these conditions the loss of pillar coal caused by falls of roof is necessarily large.

The Alabama mines, with the single exception of the Montavallo mine, where a change is being made to the long-wall system, are worked on the room-and-pillar system. The larger operations are in the neighborhood of Birmingham, and in this district the carboniferous measures are tilted and broken to a great extent. This condition affects the roof of the coal under cover for a considerable distance. Both top and bottom are of variable character.

In the larger operations at least, the triple entry system is used. Commencing at a distance of 800 feet from the surface, cross entries are usually driven about 350 feet apart. Until the entries have been driven a few hundred feet, it is not possible to determine whether they should be narrow or wide enough to provide storage for the impurities of the bed and the brushing of the roof. Rooms are generally opened narrow also, (30 feet wide) with 25-foot room pillars, until it is determined whether the character of the overlying strata and of the floor will permit the working of wider rooms.

Probably 75 per cent of the large operators in the district have adopted the plan of immediate pillar drawing in preference to that of driving the narrow work to the limit and pulling the pillars upon the retreat. In a number of instances, rooms are driven 40 feet wide with 30-foot pillars and are worked for a distance of 300 feet, or to the entry above; then a cut is taken across the end of the pillar, and the pillar is drawn back to the entry stump. When the room pillars are drawn on the advance, there is no difficulty in getting room stumps and air-course pillars after the entry work is complete.

Strong estimates the recovery in mines operated by the larger corporations to be from 87 to 90 per cent. Priestly Toulmin, another operator,* confirms these values by stating that the average extraction in Alabama is not less than 75 per cent and not more than 80 per cent, so that possibly 77½ per cent would be a fair value. In some mines the extraction is less than 60 per cent, and in others it is more than 95 per cent.

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* Personal Communication.

C. F. DeBardeleben* furnishes the following estimates:

At one operation where 768.6 acres had been worked over, assuming the average thickness of coal to be 4 feet and that 25 cubic feet of coal in place make a ton, the coal available was 5,356,834 tons

Tons mined	3,028,960
Extraction	56.5 per cent

Assuming that 12.5 per cent more will be obtained from pillars, the extraction will amount to about 63 per cent. At another operation where the average thickness is 5 feet and 316.6 acres have been worked over, the coal available was .... 2,758,219 tons.

Tons produced	1,847,582
Extraction	67 per cent

Assuming a probable extraction of pillars, the final recovery will amount to about 70 per cent. At a third mine, where conditions were favorable because of level coal and the absence of gas, 25-foot rooms were driven on 75-foot centers; the pillar was drawn back half way to the entry as soon as a room was finished, the entry pillars and room stumps being extracted when the entry was abandoned. Under these conditions the recovery was about 85 per cent.

C. H. Nesbitt, Chief Mine Inspector of Alabama, estimates the average recovery to be 80 per cent, the highest percentages being reached in the Pratt and Montavallo beds.† There has been great improvement in the percentage of extraction in the past twenty years, and even in the past ten years. This improvement has been due largely to more nearly complete and accurate mapping, and to more improved and effective methods of controlling the water in slope and shaft mines.

30. *Indiana.*—Almost no information has been available concerning the percentage of coal extracted in Indiana mines. W. M. Zeller † reports that the extraction in the Brazil district is probably about 60 per cent. This estimate agrees fairly well with the estimates of operators in southern Illinois, and since such estimates have been found to be too high in almost all instances, it is probable that the

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* Personal Communication.

† Personal Communication.

average extraction in Indiana, as in southern Illinois, will not exceed 50 per cent.

31. *Michigan.*—The coal beds in Michigan are irregular in extent and decrease in thickness with depth. Sometimes they are entirely cut out by erosion or replaced by sandstone and other materials. Usually the beds above the coal consist of black shale, and they are often weak. Owing to erosion, coal is sometimes found directly below clay, sand, or gravel, or below other unconsolidated rocks, where it is practically unworkable. At several mines the roof is of black bituminous limestone. In most instances the floor is of fire clay or shale, although sandstone is sometimes found. The thickness of coal varies from 2 feet, 6 inches to 3 feet, 10 inches, a fair average at Saginaw being 3 feet. In the Saginaw Valley the surface is level.*

R. M. Randall states † that the first company in the district operated within the city limits of Saginaw, and that because of the necessity of leaving pillars to protect the surface the recovery was only about 68 to 70 per cent. At present this company is operating in farming districts where it is not necessary to maintain the surface; and the recovery, within the last five years, has been about 90 per cent. The room-and-pillar system is used with rooms projected 40-feet wide on 50-foot centers and driven 150 feet, but the actual dimensions vary according to the conditions of the roof. Short-wall machines are used for undercutting. It is estimated that 75 per cent of the room pillars and 95 per cent of the entry pillars are recovered, and that the extraction on the advance is 70 per cent. The conditions at the old and at the new mines have been so different that it is impossible to give an average value for the extraction, but it is believed that the extraction in the new mines in the area actually worked will be from 85 to 90 per cent.

32. *Iowa.*—The physical conditions in the Iowa coal field are not uniform. The cover ranges in thickness from a few inches to 300 feet, and consists of the coal measure beds and glacial drift, the latter commonly constituting the larger part of the thickness. The workable coal beds generally have a top of draw shale varying

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* Lane, A. C., Mich. Geol. Sur., Vol. 8; Mines and Minerals, Vol. 23, p. 148.
† Personal Communication.

from a few inches to two feet in thickness. Above this is a black shale, or sometimes a bituminous or argillaceous limestone. In the latter case this rock is often strong enough to permit a reduction in the amount of timber used and thereby to facilitate mining. This condition makes it possible in Appanoose County to work a bed in which the thickness of clean coal is only about 25 to 27 inches. The bottom everywhere consists of plastic fire clay, and in the Appanoose field the undercutting is done in this clay. Its occurrence is frequently the cause of creep in room-and-pillar mines. The most unfavorable conditions are found through the northern end of the Iowa coal fields.

Even in single districts the percentage of extraction varies between wide limits. The maximum extraction, estimated at 90 per cent or more, is reached in the Appanoose field, where the long-wall system is employed. An operator* familiar with conditions in these long-wall mines says that the extraction is complete, but that it is less than the previously calculated amount of coal in the ground because of the presence of faults.

In the room-and-pillar districts the extraction rarely if ever exceeds 75 per cent, and under especially bad conditions of bottom and top with an abundance of water, it may not exceed 50 per cent. Probably a fair average of recovery for the state is 70 per cent. The percentages given refer only to the bed mined and to the area of actual mining operations. When larger areas are considered, the percentage of recovery is less because of the loss of considerable quantities of coal through lack of coöperation between owners, a loss estimated to be at least 10 per cent.†

The engineer‡ of one of the operating companies says that in the room-and-pillar mines with which he is familiar the recovery will average about 75 per cent, and that a recovery of 80 per cent is expected in the newer mines.

33. *Missouri.*—The coal fields of Missouri may be roughly divided into three districts, the first district lying near the middle of the state in Macon and Randolph Counties where operations are conducted on the room-and-pillar method, the second district farther

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* Taylor, H. N., Personal Communication.

† Beyer, Professor S. W., Personal Communication.

‡ Jorgensen, F. F., Personal Communication.

west along the Missouri River in the vicinity of Lexington where operations are conducted on the long-wall system, and the third district in the southwestern part of the state, where the conditions are similar to those of southeastern Kansas and northeastern Oklahoma and where the room-and-pillar system has generally been followed. Recently a considerable quantity of coal has been obtained in the southwest district by stripping.

In Randolph and Macon Counties, in the neighborhood of Bevier, the coal is considerably broken by faults and horsebacks, and recovery does not exceed 50 per cent.* In the long-wall district the extraction in the area worked out is practically complete, but most of the operations are conducted on a small scale and no estimates covering the probable extraction over the whole area are available.

In the southwestern part of the state the continuity of the coal is considerably broken by horsebacks, as in the neighboring parts of Kansas and Oklahoma. Mining methods have not been highly developed, and no great attention has been paid to completeness of extraction. It is not probable that the extraction in this district, within the areas worked, will be more than 50 per cent.

34. *Arkansas.*—Steel says† that the ordinary waste of coal in Arkansas is unusually great even for this country, a fact to be accounted for partly by unfavorable geological conditions. In addition to the wastes common to all coal producing states there are others due to local geological and physical conditions, which Steel considers unusually unfavorable in Arkansas.

There is considerable loss because of irregularities of entries, due to the varying dip of the bed. Entries which are turned from the slope at standard distances measured along the coal seam will have variable and perhaps severe grades if they are driven straight or will be very crooked if they are driven on grade. If the dip increases and the entries are driven on grade, the distance between entries decreases and sometimes the rooms between entries become so short that the entry from which they are turned is discontinued; then rooms from the entry below are driven long enough to take out all the coal, or part of it, which would have been taken out through the intermediate entry. Sometimes the length of rooms necessary to extract all the coal would be so great that part of it is left. Sometimes this is won through the upper entry, and sometimes it is lost.

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* Taylor, H. N., Personal Communication.

† Steel, A. A., "Coal Mining in Arkansas," Ark Geol. Sur.

There are also losses due to irregularities in the coal, and entries are frequently not extended through areas of low coal to get the good coal lying beyond. Areas of thin coal are commonly abandoned. The losses due to thin or poor coal are greater perhaps in Arkansas than in other states, because the dip of the beds makes the driving of entries around these poor areas considerably more expensive, and it is not profitable to take out good coal lying beyond poor coal unless the area of the good coal is large. There is often considerable loss from the abandonment of parts of beds. In many places the different benches of thick beds are separated by thick partings; if a single bench is thick enough to mine, it is worked separately, and sometimes the bench above it or below it is lost. The loss of coal in this form, though not so great as formerly, is probably greater in Arkansas than in any other state, with the possible exception of Colorado. Loss due to the need of protecting the surface is not serious, because the value of the surface is low, and the rough topography insures good drainage.

H. Denman,* an operator familiar with the district, expresses the opinion that the recovery in both Arkansas and Oklahoma does not exceed 50 per cent. In certain portions of a mine the recovery may be as high as 70 per cent, but he believes if the whole area of the mine is considered, the percentage of extraction will not, in any case, exceed 55 per cent. These statements are applicable to both Arkansas and the neighboring Oklahoma district, as the same system of mining is used in both.

The system of mining in the Arkansas-Oklahoma field is practically the same that was used when the field was first opened about forty years ago. There is no systematic attempt at laying out mines with the view of drawing pillars, but the general plan is to get as much coal as possible in the first working and to abandon the remainder. There is one mine in which an attempt is being made to plan the work so as to obtain the pillar coal, but this attempt is so recent that it is impossible to foretell the degree of its success. The widths of rooms and pillars are influenced by the charges for narrow work and for yardage, which are so high that neither narrow rooms nor long break-throughs can be driven. At present the average room neck is about 10 feet long, and if longer necks could be driven without increased cost, it might be possible to prevent squeezing of the entry and to obtain a considerable amount of coal from the entry pillars, but the yardage cost is so high that this procedure seems unprofitable.

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* Personal Communication.

35. *Kansas.*—The coal produced in Kansas comes from three districts, the one in the southeastern corner of the state being by far the most important. The others are the Leavenworth and the Osage districts. The Leavenworth district lies in the northeastern part of the state and may be considered as connected with the district of northwest Missouri, although the strata dip toward the west and the coal is found at greater depths in Kansas than in Missouri. All operations in the Leavenworth district are on the long-wall system, and the extraction, in the areas mined out, is practically complete. Coal in this district ranges from about 19 to about 24 inches in thickness. The depth is about 700 feet. A 3-foot bed lying at a depth of 1,000 feet was found at Atchison about ten years ago and was worked by the long-wall method, but the work was not commercially successful and was abandoned. The Osage district lies to the south of Topeka and is not important commercially. The coal is about 20 inches thick, and is mined entirely by the long-wall method. The extraction is practically complete within the area mined out. This is the thinnest bed of bituminous coal worked in the United States.

In the southeastern district of the state the coal beds lie on the west slope of the Ozark uplift and dip toward the west and northwest. The beds contain numerous horsebacks which interefere with systematic mining. The room-and-pillar method is followed, and little attempt is made to extract pillar coal. Practically all coal in Kansas, except that produced by the long-wall method and by stripping, is shot from the solid, a method which unquestionably leads to the production of small coal, especially where the holes are greatly overcharged as they usually are. The recovery is in the neighborhood of 50 per cent, although it may sometimes be greater in limited areas, because the horsebacks may be made to serve as pillars. H. N. Taylor, in a personal communication, confirms this estimate of extraction. He says that in places a considerable loss is experienced, because the rate for mining low coal is so high as to be considered prohibitive, and even if the rate is paid it is difficult to get men to work the low coal. A. C. Terrill reports* that the closest estimates of those most familiar with conditions place the recovery at about 50 per cent.

The approaching exhaustion of the shallower mines has necessitated the working of the northern part of the district where the cover is about 250 feet thick. At least one of the operators desired to work these deeper mines by the panel method, but it has not as yet been found possible to reach satisfactory arrangements with the mine workers. Since a makeshift adopted to prevent the spread of squeezes leaves two rooms out of seven unworked, the recovery has been reduced about 1,000 tons per acre. The operators still hope that they may be able to introduce the panel system and thus materially increase the recovery.†

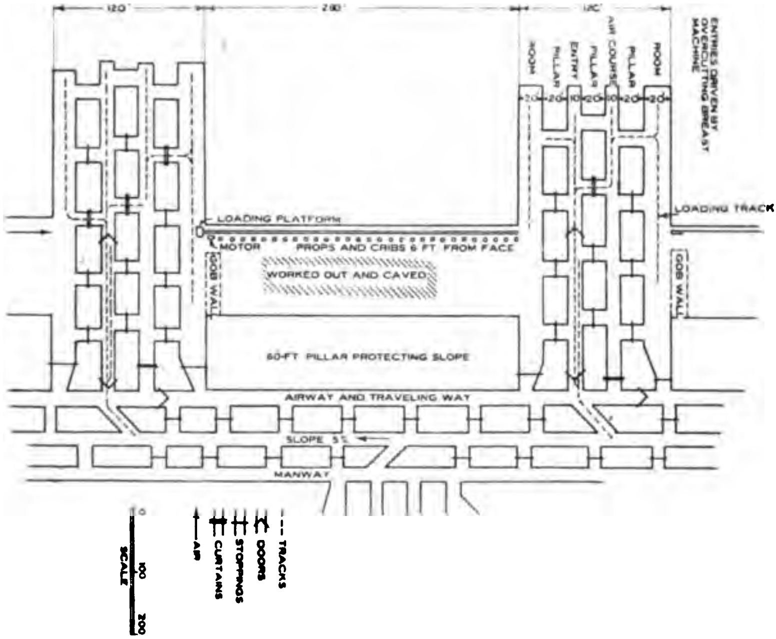

FIG. 38. PANEL LONG-WALL IN OKLAHOMA

In recent years a large amount of coal has been taken from this district and from the neighboring region in Missouri by extensive stripping operations. In the area worked over, the extraction by this method is practically complete.

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* Personal Communication.

† Taylor, H. N., Personal Communication.

36. *Oklahoma.*—In Oklahoma the coal is produced largely by individual operators, the land being owned by the Indian nations and leased to operators in small tracts.* Elaborate plans for mining are not to be expected under these conditions. Elliot estimated that the recovery of the entire bed worked was not more than 55 per cent. He said that the low percentage of recovery was due to the extravagant system of room-and-pillar mining adopted, and that this system could not be changed because of unfavorable labor conditions.

The Rock Island Coal Mining Company obtains an extraction of 48.18 per cent in the McAlester district. In the Hartshorne district this company has five mines, their percentages of extraction being 56.6, 52.6, 55.0, 51.5, and 47.8, respectively. The average percentage of extraction at these five mines is 52.7 and the average of all the mines of the company in Oklahoma is 51.8. †

This company is now trying a panel long-wall plan (Fig. 38) with the hope of increasing the extraction from about 57 to about 70 per cent. The coal is about 3 feet, 4 inches thick, and dips from 5 to 8 degrees. The working face is parallel with the dip. The roof along the face was at first supported by cribs built of 8-inch by 8-inch timbers about 4 feet long and these cribs were withdrawn and moved forward as the face advanced, the roof being allowed to fall. A row of props was also used to support the top above a conveyor used for carrying the coal along the face. At present the use of cribs has been discontinued, except along the ribs of the entries, and 10-inch by 10-inch props are used to support the roof. These are drawn and reset as the face advances. The necessity of using props on both sides of the conveyor constitutes one of the difficulties of the operation. The roof breaks as the face advances. There seems to be no great difficulty in the use of undercutting machines, but sometimes the coal falls too soon for convenience in loading, and large lumps clog the conveyor. While this operation must still be considered in the experimental stage, the working face has been advanced about 130 feet without serious difficulty. It is planned that the pillars flanking the long-wall panel, left during the advance, shall be taken out on the retreat.

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* Elliot, James, "Conditions of the Coal Mining Industry of Oklahoma," Proc. Amer. Min. Cong., p. 221, 1911.

† Scholz, Carl, Personal Communication.

37. *Texas.*—There are three bituminous coal fields and one lignite field in Texas. In one bitumionus field in the north central portion of the state, practically all the mines are operated on the long-wall plan, and the recovery is nearly complete. The two other bituminous fields are located in the southwestern part of the state along the Rio Grande. In this district the mines are operated on the room-and-pillar plan, and the recovery is said to be about 75 per cent. The lignite field extends entirely across the state from the northeast corner in a southwesterly direction to the Rio Grande. All lignite mines are worked on the room-and-pillar method, and the recovery varies greatly in different parts of the state, but 75 per cent is probably the average.*

38. *North Dakota.*—J. W. Bliss, State Engineer, estimates that the recovery in the coal mines of North Dakota is between 70 and 75 per cent. The manager of one mine claims a recovery of about 85 per cent. †

39. *Colorado.*—The principal producing districts of Colorado are the bituminous district in the southeastern part of the state, near Trinidad, and the lignite district just east of the mountains in the northern part of the state. The bituminous district is the more important. In the Trinidad district‡ the average thickness of the coal is about 6 feet. The top is strong, and the bottom is weak. Entries are driven to a fixed boundary, and the rooms which are needed to supply enough coal to keep the driver busy are turned. When the boundary is reached, rooms are turned at the inby end of the entry, and pillar drawing is commenced as soon as the rooms reach their limits. Nearly all the coal is taken out on the retreat. Rooms have a maximum width of 18 feet, and room pillars are 32 feet wide. All work is done with picks. The coal is soft and occasionally the pillars crush, but most of the difficulty encountered is due to heaving of the bottom. The cover averages more than 600

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* Gentry, B. S., State Inspector of Mines, Personal Communication.

† Personal Communication.

‡ Weitzel, E. H., Personal Communication.

feet in thickness. The output from mines in this district is used largely in connection with steel making, and operations are very regular, most of the mines being worked every day in the year. The recovery at a typical mine in this district, calculated for operations over a period of several years, is 87.2 per cent.

In the domestic coal district in the neighborhood of Walsenburg, the average thickness of coal is 5 feet. It is stronger than the coal in the Trinidad district, and a squeeze is very unusual. The cover averages about 400 feet in thickness. As a rule the bottom in this district is stronger than the top, and little difficulty is experienced from heaving. The work is less regular than in the Trinidad district, although it is fairly regular except in March and April when the mines are usually worked about half time. Rooms are driven 25 feet wide on 50-foot centers. There is little difficulty in drawing pillars. The tendency in these districts has been to drive narrow rooms and to leave wide pillars, and this has assisted in increasing the percentage of recovery. The extraction in a typical mine in this district, calculated for operations over a period of several years, is 91.7 per cent. The chief engineer* of another company operating in this same district believes the extraction in certain portions of the mines of his company will reach 80 per cent. In the Canyon district the long-wall system is used, and the recovery is nearly complete.

40. *New Mexico.*—No information is available concerning the percentages of extraction in New Mexico.

41. *Utah.*—The principal coal fields of Utah are located in Carbon County.† The main coal horizon has from two to four workable beds, from 5 to 28 feet in thickness. The main workable bed, known as the Castle Gate, varies in thickness from 5 to 20 feet, and rests on a massive close-grained sandstone. The problem presented by these deposits is one of mining thick seams, comparatively level or slightly inclined. Formerly some seams 4½ to 8 feet in thickness were worked, but at present most of the mining is done in seams varying from 8 to 28 feet in thickness. The physical features to be taken into consideration in this district are: the number of workable seams, the thickness of seams and their relation to one another,

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* Personal Communication.

† Watts, A. C., "Coal Mining Methods in Utah," Coal Age, Vol. 10, p. 214 and p. 258; and Personal Communication.

the character of the coal, the dip of seams, the character of roof and floor, cover, faults, dikes, wants, the flow of water, sometimes gas, and the burned-out coal beds in place with their residual heat.

Throughout most of the fields there are at least two workable seams, and these are generally found in one coal horizon. In several sections, however, there are three and sometimes four workable seams 5 feet or more in thickness. The distances between these seams vary considerably, so that in some sections there are no unusual problems involved while in others two or more workable seams are found with so little intervening strata that the problem of successful extraction has not yet been solved. There may be, for example, an 8- to 14-foot seam underlying a 6- to 10-foot seam with about 200 feet of intervening strata; in another instance a 5- to 8-foot bed lies 60 feet below one 5 to 11 feet thick, and this lies from 12 to 20 feet below a 22-foot seam, which in turn lies 30 feet below a 6-foot seam. In still another instance an 11-foot bed is found from 3 to 40 feet below a 6-foot bed. There is considerable variation in the physical characteristics of the beds, some being hard and brittle and others tough. In some instances the cleavage is good, while in others it is not pronounced. Almost without exception the coals are hard to cut, and some are hard to shoot. The average dip does not exceed 10 per cent, and in some places the beds are practically flat. As a rule the floor consists of hard smooth sandstone from which the coal parts rather readily. In many cases the roof is of shale varying in thickness from a few inches to several feet. Where a sandstone roof is found, it is generally too hard to break for easy mining. In some places the cover is more than 2,000 feet thick, and there are only a few localities in which it is less than 1,000 feet in thickness. This heavy cover makes the mining of these flat thick seams a serious problem in itself, but the additional complication of great irregularity in depth and the unyielding qualities of the thick beds of overlying sandstone make the problem still more serious. A condition which modifies, at least locally, the laying out and working of a mine is the fact that near the outcrop there are sometimes found large areas of burned coal. These sometimes extend 2,500 feet in from the outcrop. Mining in burned areas is often dangerous, if the burning has been at the top, because of the disintegration of the roof.

With one exception all the mines of the district are opened from the outcrop by means of slopes of drifts. Where conditions of topography and property permit, main slopes are driven directly on the pitch of the seams. All mining is by the room-and-pillar method. An attempt to use the long-wall method in one case failed because of the unyielding nature of the roof. The double-entry system is almost universal, although in one case a triple entry is used, and in some cases the double-entry system has been so modified by the connection of the first rooms on the cross entries that it has become practically a 4-entry system. In the earlier workings rooms were turned from the cross entries as these were driven, but the system resulted in the occurrence of bounces, which seem to take the place of the squeezes that occur with more yielding materials. In later operations the panel system has been used, and the pillars are drawn on the retreat.

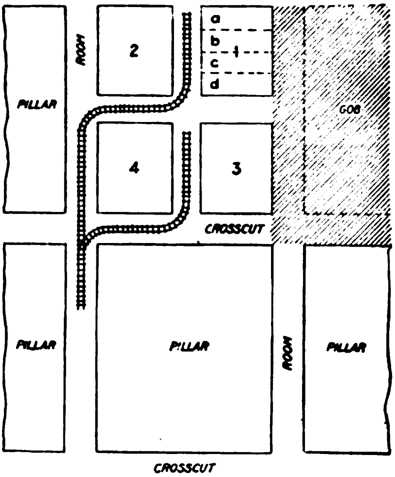


Fig. 39. Pillar Drawing in Utah

Methods of drawing pillars are of particular interest, since they show how almost complete extraction can be attained under conditions which seem unfavorable. These are described by Watts substantially as follows:

In one method (Fig. 39), the block at the end of the pillar on the inby side of the cross-cut is divided by another cross-cut driven through its center, and from the center of this new cross-cut a narrow room which splits the stump into two parts is driven to the gob. The upper inside stump is taken out first by slices beginning at the inby end; then the remaining stump is removed. The lower half of the original block or pillar meanwhile is split by a narrow road, and the process is thus continued down the pillar, each block being divided into four parts.

Pillar drawing in a flat seam 12 to 14 feet thick under a cover 800 feet thick and under a roof which broke fairly well when posts were removed has been successfully accomplished by the following method: 20-foot rooms were driven with 50-foot pillars (Fig. 40), and a cross-cut was driven through the pillar; thus a 30-foot stump was left. This 30- by 50-foot stump was then split by a 12-foot room which left a 24- by 30-foot stump next to the room and a 14- by 30-foot stump on the other side. The latter stump was then taken out in slices which begin at the gob, and the roof was supported by props set every 4 feet. The coal was undercut by hand and shot with black powder. When this block had been removed, the track was taken up, and all props were drawn except a row adjacent to the rib of block No. 3. These blocks were numberd in the order of their extraction, 1, 2, 3, and 4. Block No. 3 was then taken out from the

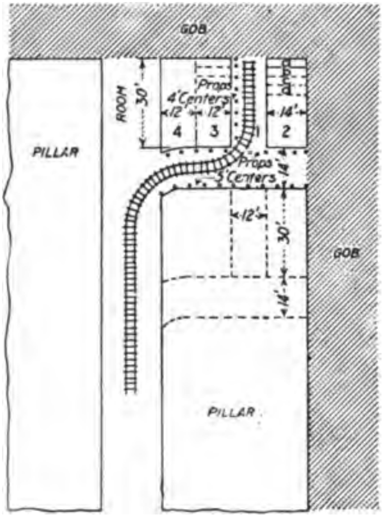


FIG. 40. PILLAR DRAWING IN UTAH

cross-cut to the gob, track was laid in the space, and block No. 4 was taken out in the reverse direction, that is, beginning at the gob. In this mine 3 to 6 feet of top coal are left up to protect the roof on the advance, but this coal is taken down on the retreat. When cross-cuts are made in pillars preparatory to drawing them the whole height of the seam is taken.

Pillar drawing in 16-foot coal with a cover of 400 to 1,000 feet, with no top seam and with the roof breaking well when props are drawn, is accomplished as follows: Rooms about 400 feet long are driven straight up the pitch which averages about 10 per cent. Pillars are drawn on the retreat, and the line of break is kept at an angle of 45 degrees with the entry; thus work is done on six or seven pillars at a time. Rooms are about 20 feet wide, and pillars are 50 feet wide. A cross-cut is driven through the pillar 30 to 35 feet from the end (Fig. 41); thus a block about 25 to 30 feet by 50 feet is cut off. This block is then split by a room about 12 feet wide. Blocks 1 and 2 are drawn by slicing which begins at the end next to the gob. The top is supported by means of props at 4-foot intervals, and after the two blocks have been removed and the track has been taken out, these props are pulled, and the area is allowed to cave. The track is then laid in the main room, and blocks 3 and 4 are taken out by

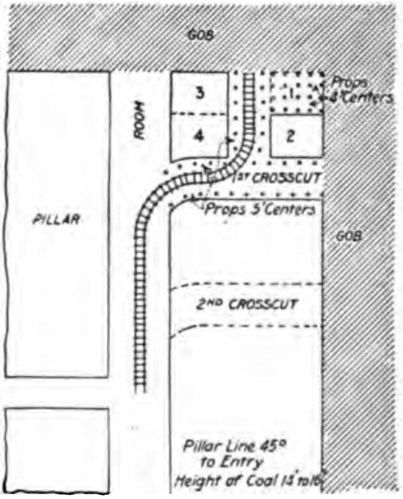
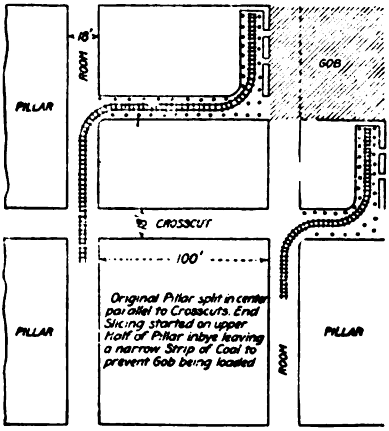


FIG. 41. PILLAR DRAWING IN UTAH

end slicing from the room; then the track is taken out, and the props are pulled. Another cross-cut meanwhile has been made through the pillar nearer the entry, and a room has been driven through the stump so that by the time the first stumps have been extracted, work is being begun on the lower stumps.

In this mine the size and the systematic placing of props have an important bearing on the successful recovery of the pillars. Until heavy pine props were used, trouble was likely to occur at any time. Now props as large as 10 inches in diameter at the small end with correspondingly heavy caps and, in some places, cross bars are used. Props are set at regular distances. Many of these props are recovered and re-used three or four times. By the adoption of this method, the safety factor is largely increased, the percentage of recovery is greater, and the product is of better quality. In some cases it is customary to mark pillars at regular distances so that the mine foreman or pillar boss may easily determine the progress of the pillar work daily and may keep the ends of the stumps in proper alignment.



Original Pillar split in center parallel to Crosscuts. End Slicing started on upper Half of Pillar inbye leaving a narrow Strip of Coal to prevent Gob being loaded

FIG. 42. PILLAR DRAWING IN UTAH

Another method of pillar drawing sometimes used is similar to that last mentioned, although the stump left is a little shorter. This stump is then split into quarters, and the work of extraction proceeds from the cross-cut toward the gob, a thin section of coal being left to the last around the edges of the block to prevent the mixture of fallen roof with clean coal.

A plan which has proved satisfactory, partly because it does not split the pillar into many small stumps, is illustrated by Fig. 42. The original block of pillar coal 100 feet wide by 120 feet long is divided into equal parts by a cross-cut, and the upper half is taken out by slicing beginning at the inner side, a curtain of coal being left to prevent the loading of gob, and one or two rows of props being put along the side of the coal. Before the track and props are pulled, most of this section of coal is loaded out. In this case the rooms are 18 feet wide and the pillars 100 feet wide.

Little information is available regarding the percentage of extraction in Utah. There is probably only one mine in the state which has been worked out, and no reliable information can be obtained concerning this mine. It is believed, however, that the extraction was probably about 75 per cent. In coal 12 to 16 feet thick and under cover varying from 200 to 2,000 feet an extraction as high as 90 per cent has been made, if marketable coal alone is considered. If all the coal in the bed is considered, the recovery is about 80 per cent. In beds ranging from 15 to 30 feet in thickness, retreating work has hardly been started so that no information on total recovery is available. It is possible that it will be rather low. It could be made higher if the filling method could be used, but the price of coal does not warrant the use of this method.

A condition largely influencing the percentage of extraction is the presence of more than one workable seam with little intervening material. Under present conditions the percentage of extraction from an area containing seams with 3 to 12 feet of intervening rock is at best only 65 per cent of all the coal. In one mine an attempt was made to take out the coal from two beds, the lower being 11 feet thick and the upper 5 to 6 feet thick with intervening rock 2½ to 12 feet thick. The workings were in the lower bed, and frequently the roof caved as soon as the pillars were drawn and practically all the upper seam was lost.

A. B. Apperson* gives the percentage of extraction in two mines as nearly 95 per cent of the total seam, while the extraction at another mine is about 85 per cent. At the mines yielding the lower percentage of extraction, the cover is about 800 feet. At one of the mines yielding the higher percentage of extraction, pillar drawing was commenced at the middle of the mine under a cover of approximately 1,700 feet. A good break is obtained about 50 feet behind the pillar extending the full length of line. Only small areas have been worked out in these mines.

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* Personal Communication.

42. *Washington.*—No reliable information is available concerning the percentage of extraction in Washington. Conditions are somewhat unusual in that most of the coal has been badly folded and faulted and consequently crushed, and the deposits have been steeply tilted.* It is impossible to separate the refuse in the mines, and a large percentage of it has to be washed.

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* Daniel, Professor Jos., Personal Communication.

APPENDIX

DEVELOPMENT OF MINING METHODS IN ENGLAND AND ON THE CONTINENT

43. *Brief History of Coal Mining Practice in England.*—It is interesting to review briefly the history of the coal mining methods of England, because the mining methods employed in this country are largely applications of methods developed in England and brought over by miners.

The many methods of obtaining coal may be grouped on the basis of recovery under two main headings: one in which the whole of the coal seam is taken out in the first working, and another in which only a part of the seam is removed in the first working. These may be called the no-pillar, or long-wall, system and the pillar system.

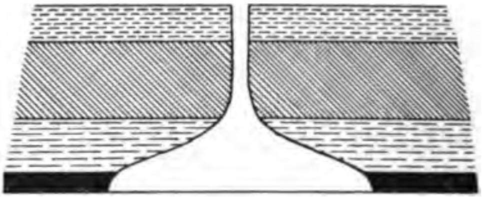

FIG. 43. BELL PIT

The earliest mining was naturally done on the outcrop of the seams, and as this practice became difficult or impossible, the use of "bell-pits" (Fig. 43) was developed. These were holes or shafts, from 3 to 4 feet in diameter, which were sunk through the shallow overburden near the outcrop and widened out at the bottom in order to allow the excavation of as much coal as possible without permitting the roof to fall in. It was of course impossible to extract much coal from a pit of this kind, and in order to obtain the coal even from a small area it was necessary to dig a large number of pits. This method was gradually abandoned, and the coal was worked by means of galleries driven out from the bottom of the shaft, usually in an unsystematic manner; thus began the use of pillars to sustain the roof. The driving of galleries permitted the working of much greater areas than could be reached from the bell-pits; however, no areas of more than a few acres were worked from one shaft, nor were systematic ventilation and regularity in laying out the workings introduced until the exhaustion of the shallow coal made necessary a study of methods to be employed in deeper workings. Until the introduction of the Newcomen engine, when pumping by steam power became possible, shafts were rarely as deep as 200 feet; they were 7 or 8 feet in diameter, and the area worked from one shaft was seldom more than 600 feet in radius.*

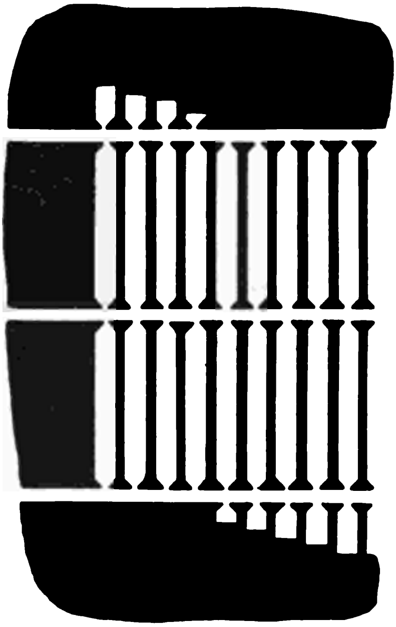

FIG. 44. BORD-AND-PILLAR

The structure of many coal seams is such that there are two directions, determined by the cleat of the coal, in which the seams can be most easily worked. The direction at right angles to the face cleats is known as "bordway," while the other direction approximately at right angles to the first is known as "headway." The excavations made in a direction at right angles to the principal or face cleats were called bords, and, as the coal was most easily taken out in this direction, these excavations were made wider than the connecting passages or headways. The coal left in place to sustain the roof was called pillars; thus originated the term, "bord-and-pillar" method, which in its various developments is commonly known in this country as the room-and-pillar method. This method was developed in different forms in England and was variously called "bord-and-pillar," "bord-and-wall," "post-and-stall," and "stoop-and-room."

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* Bulman and Redmayne, "Colliery Working and Management," p. 3, 1906.

In early times the "pillars" were probably made very small and square measuring from 3 to 6 feet each way. In the eighteenth century the bords were usually made 9 feet wide and the pillars 12 feet wide, though they were of course irregular. The bords were commonly widened out between the headways (Fig. 44), and the pillars were thus gouged to as great an extent as was considered safe, it being desirable, in view of the comparatively small area which could be reached from a single shaft and in view also of the inadequate ventilation, to extract as much coal as possible within the area worked. This method of working was essentially wasteful as not much more than 50 per cent of the coal was obtained, and since the pillars left were unable to bear the weight of the cover, they were soon crushed and further working was made impossible.* Possibly a larger percentage of coal was taken out in some places as Redmayne † says it was rare that more than 65 per cent of the available area could be

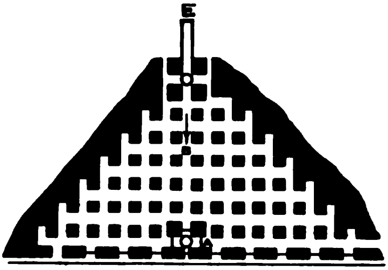

FIG. 45. STOOP-AND-ROOM

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* Boulton, W. S., "Practical Coal Mining," Vol. 1, p. 296.

† Redmayne, R. A. S., "Modern Practice in Mining," Vol. 3, p. 82.

extracted; but he refers to John Buddle as saying, at a somewhat later period when the pillars were 18 by 66 feet, that not more than 45½ per cent of the contents of a fiery seam could be obtained under any method of working then known. It was not until much later that the exhaustion of the most easily worked deposits directed attention to the desirability of higher extraction.

A system with square pillars and working places of almost uniform width (Fig. 45) has continued in common use at Whitehaven and in Scotland down to the present time. In the north of England the pillars were usually oblong, probably because the highly developed face cleat of the coal made the extraction in one direction much easier than in others.* The lengthening of the pillars reached its greatest extent in South Wales where cross holing was so little employed as scarcely to form a part of the system of working.

The date at which the extraction of pillar coal was begun is not known, but it seems certain that pillars were removed in the north of England before 1740. The following statement† is made concerning the removal of pillars:— "The documentary evidence cited goes to show that, previous to 1708, the general practice was to leave small pillars of coal standing for the support of the roof; 30 years later pillars were being partially, sometimes entirely, removed; and during the remainder of that century, in mines free from gas, a second working of the pillars was frequently carried out. In the deeper and fiery collieries, which began to be developed about the middle of the eighteenth century, the risk of creep as well as of gas explosions prevented the removal of the pillars. The invention of the safety lamp, improvements in ventilation, and the formation of much larger pillars in the first working . . . . were introduced during the first 30 to 40 years of the present (nineteenth) century . . . . which enabled the pillars to be removed in a second working."

Concerning the size of pillars, Jars, a French engineer who published "Voyages Métallurgiques" in 1774, says in "A Journey Through the North of England," that underground pillars of coal were made from 39 to 54 feet square, and that working places were from 5 to 16 feet wide. At this time the pillars were left until all the coal was exhausted. Another traveler who made a tour of Scotland in

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* Galloway, R. L., "Annals of Coal Mining and the Coal Trade," p. 181, 1898.

† Bulman and Redmayne, "Colliery Working and Management," p. 14, 1906.

1772 said that pillars 45 feet square were left and that not more than one-third of the coal was worked.*

The extraction of a portion of the pillars in gassy mines by a second working was just beginning to be a regular part of the bord-and-pillar system at this period. It could, however, be effected only in a very incomplete manner so long as the miners had to depend upon candles and steel mills for light. At this time also the extensive adoption of the long-wall system began.†

In the early part of the nineteenth century little change seems to have been made in the size of pillars used in the Newcastle district, according to a statement of an author who speaks of them as being 60 by 27 feet or, in some instances, 27 feet square. About this time the drawing of pillars seems to have become common in Northumberland, as Mackenzie, who wrote a "View of Northumberland" in 1825, speaks of the mode of working coal as being much improved in the last few years. He says (second edition, page 90), "from seven-eights to nine-tenths of the coal is at present raised, whilst formerly but one-half, and frequently less, was all that could be obtained." No doubt this statement refers to the general practice of removing pillars, which had been made practicable in gassy mines by the invention of the Davy lamp.

Conflicts of interests between coal producers and owners of the surface are of early record. It was, of course, the desire of the colliers to remove as much of the coal as possible, even where the surface was supposed to be maintained, and the result of making pillars too small was subsidence. There is probably no definite record of the first occurrence of subsidence, but one of the earliest mining leases written in the English language, dated 1447, indicates that it was the custom to leave pillars to sustain the surface and that subsidence had already taken place.‡

In the latter part of the eighteenth century the working of pillars in a fiery mine, such as Wallsend Colliery, was not considered practicable, and only about 39 per cent of the coal was obtained while 61 per cent was permanently lost. This coal was at a depth of 600 feet, and the workings represent the best practice of the bord-and-pillar system at that period.¶

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* Galloway, R. L., "Annals of Coal Mining and the Coal Trade," p. 353, 1898.

† Ibid, p. 363.

‡ Ibid, p. 69.

¶ Ibid. p. 298.

Until about the end of the eighteenth century an extraction of 45.5 per cent was considered the maximum which could be obtained in the deep collieries of the Tyne.* The first person to offer a partial remedy for this very unsatisfactory condition was Thomas Barnes, viewer of Walker Colliery, who projected a scheme in 1795 for recovering a portion of the pillars without causing loss of the mine. This system provided for dividing the workings into small sections of 10 to 20 acres and isolating these with artificial barriers formed by filling the excavated spaces with stones and refuse for a breadth of 120 to 150 feet. By this method one-half of alternate pillars, or one-quarter of the remaining coal, was removed, and the percentage of extraction was increased from about 39 to about 54 per cent. Wherever pillars were thus removed, a squeeze was brought on, but the barriers kept it from spreading. This method proved to be successful, and it was adopted at other collieries.

Probably about this time the pillars at Wallsend Colliery were left larger as a preparatory step toward a second working. Buddle said that after about one-third of the colliery had been worked by means of 36-foot winnings (12 feet to the bord, 24 feet to the wall or pillar) in which no more coal was left in pillars than was considered sufficient to support the roof, the size of the winnings was increased to 45 feet (15 feet to the bord and 30 feet to the pillar). "This change of size," he said, "was not made for the purpose of obtaining a greater produce in the first working of the seam. But the notion of the future working of the pillars then began to be entertained, and the increased size of the winnings was considered a more favorable apportionment of the excavation and pillar for the attainment of this object." This is the first record found of a second working in the deep Tyne Collieries.† Pillars seem to have been worked in the northern part of England about the middle of the eighteenth century.

44. *Ventilation.*—The distance to which workings could be driven and the extent to which pillars could be drawn, especially in gassy mines, were found to depend largely upon ventilation. In the latter half of the eighteenth century improvements in ventilation, which had been used earlier in the Cumberland field, were introduced into

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* "Trans. Nat. Hist. Soc. of Northumberland," Vol. 2, p. 323.

† Galloway, R. L., "Annals of Coal Mining and the Coal Trade," pp. 315–318, 1898.

the north of England, where the frequency of explosions made better ventilation necessary. Until this time it had been considered sufficient to conduct the air current along the working face, an arrangement known as " face-airing;" consequently, the worked-out places, which were behind the miners in advancing work, were left without ventilation. As long as the extent of workings was very limited, this method was not attended with great danger; but, as the mines became deeper and more gassy, workings were made larger and the danger from this inadequate ventilation increased, because the worked-out places became magazines for the accumulation of fire damp.

The improved method of ventilation, which was known as "coursing the air," consisted in so directing the air that the whole current passed through all the openings in the mine. While this method was effective in preventing the accumulation of standing gas, it introduced a great danger in that the air took up constantly increasing quantities of gas in its passage through the mine, and, since it was constantly exposed to the lights of miners, it became dangerous in the latter part of its course. It was, moreover, constantly contaminated by the breathing of men and animals and by the smoke from the candles. This method was introduced in the north of England about 1765 or 1766, and it was about this date that the steel mill also was introduced for the purpose of giving light.* Though this method was fairly satisfactory in small mines, it was very unsatisfactory in large ones. At Walker Colliery, although the pits were only half a mile apart, the air current traversed a line exceeding thirty miles in length. At Hebburn Colliery the air course was also said to be not less than the same length. Not only was it difficult to keep the air passages open and the doors and stoppings tight, but the friction of the air limited the velocity of the ventilating current, which would have been low at best since the force causing this current was supplied only by a furnace. At this colliery the circulation of five or six thousand cubic feet of air per minute was considered sufficient, and the velocity was about three feet per second.

45. *The Panel System.*—There was great difficulty in carrying on work in the deep collieries of the North, because squeezes occurred. A method of working described as common in the North at this period consisted in having bords 12 feet wide and 24 feet apart

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*Galloway, R. L., "Annals of Coal Mining and the Coal Trade," p. 279, 1898.

connected by headways 60 feet apart, thus leaving pillars of coal 24 by 60 feet. Another size of pillar given is 30 by 72 feet.*

Early in the nineteenth century, John Buddle, Jr., who had succeeded his father as manager at Wallsend and who was responsible for important improvements in coal mining methods, devised and put into practice improved methods of working and ventilation whereby squeezes were effectually kept in check, and the existing system of ventilation was greatly improved. He effected these improvements by dividing the workings into independent districts or panels, as Barnes had done. Buddle's idea, however, was to provide for confining the movement by separating the districts or panels with barriers of solid coal left in the first working. This method was adopted in developing the Wallsend G pit in 1810. Buddle's improvement in ventilation involved dividing or splitting the current. This method of ventilation proved successful and was quickly adopted at other mines to which it could be applied, but the air currents employed were still very feeble.

From the preceding descriptions it will be seen that all the essentials of the room-and-pillar system as now practiced in this country had been developed in Great Britain prior to 1810.

46. *Square Work of South Staffordshire.*—In the Thick seam of South Staffordshire where the coal varies in thickness from 18 to 36 feet, a method which bears a close resemblance to the panel method was developed. The district had been greatly troubled with fires due to spontaneous combustion, and in order to extinguish these fires easily or to confine them within the immediate vicinity of their origin this method, known as "square work," was developed. It consists in dividing the area to be worked into a number of large chambers termed "sides-of-work," surrounded on all sides by panels of solid coal known as "fire ribs." The only openings in these panels are those necessary for the extraction of coal and for ventilation. The panels are nearly square, and from four to sixteen pillars, the number varying according to the size of the chamber, are left to support the roof. Fig. 46 shows an old form of square work. Under the system in its simple form and in the first working, only from 40 to 50 per cent of the available coal is recovered, but the larger portion of that left is recovered by second or even third

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* Ibid, p. 395.

workings carried out after the lapse of some years. The final loss in working may, therefore, not exceed 10 per cent of the available coal; the coal recovered in these later workings, however, is frequently badly crushed.*

47. *The Long-wall System.*—The other general method of coal mining, the long-wall method, has been from early times prevalent in Shropshire, from which district it has spread into others. The date of the origin of this method is doubtful, but it is said to have been in general use in the Shropshire district about the middle of the nineteenth century.†

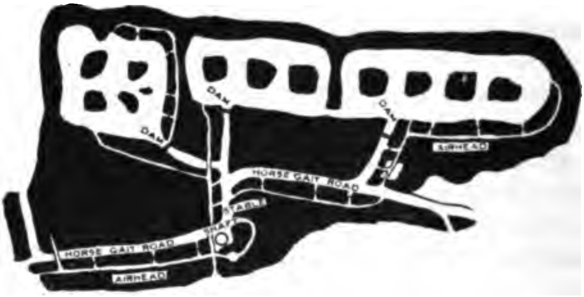

FIG. 46. OLD SQUARE WORK

The long-wall method of mining has been highly developed in England and Scotland and has been applied at greater depths and to thicker beds of coal than it has been in this country. Considered from the point of view of completeness of extraction, the system fulfills the highest requirements: It not only permits, but requires the excavation of the whole bed of coal. Whether all the coal shall be taken out of the mine depends of course on whether it is marketable.

This method of working has not been as generally applied in the United States as have the various forms of the room-and-pillar system. There are, however, certain districts in which it is used almost exclusively. Among the most prominent of these is the long-wall district of Illinois which has been described as District I in Chapter II. The other districts in which the long-wall method is used are those of northwest Missouri, northeast Kansas, the

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* Redmayne, R. A. S., "Modern Practice in Mining," Vol. 3, p. 116.

† "Trans. North of Eng. Inst. Min. Engrs.," Vol. 2, p. 261.

Osage district of Kansas, the Appanoose district of Iowa, the north-central district of Texas, and the Canyon district of Colorado. Scattering applications of the method are found elsewhere, but the physical conditions in the regions mentioned have been best suited to its use. It seems probable that the long-wall system, with modifications perhaps, will be more widely used in this country in the future.

48. *Percentage of Recovery in England.*—The methods followed in England have not been developed with the purpose of obtaining a high percentage of recovery. It was not until 1854* that a special department for collecting and publishing mineral statistics was created, and not until 1861 that any systematic estimate of coal resources was made. About this time predictions forecasting the exhaustion of the coal supply within a century caused a great disturbance, and a royal commission was appointed in 1866. A report of this commission was made public in 1871.

The part of this report which deals with waste in working is of special interest. The commission estimated the "ordinary and unavoidable loss" to be about 10 per cent, though they said, "In a large number of instances, when the system of working practiced is not suited to the peculiarities of the seams, the ordinary waste and loss amount to sometimes as much as 40 per cent." The principal part of this unavoidable waste arises from the crushing of pillars.

In addition to this unavoidable loss, there is waste or loss, variable in amount, but sometimes very great, arising from the following causes:†

(1) The leaving below ground or consuming in large heaps of small coal on the surface (presumably the loss from this source is much less at present because of the greater consumption of small sized coal, as in this country).

(2) Undercutting, often wastefully made, in good coal.

(3) The leaving, either wholly or in part, of an adjoining or neighboring bed when it becomes crushed and unworkable, because it is not wanted at the time, or because if it should be worked, the cost per ton of the coal extracted is increased.

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* Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. IX., 1905.

† Ibid, p. XXXIII.

(4) Existence of coal on properties which are too small to be worked alone or in small parts of colleries cut off by a large fault.

(5) Disputes over the cost of drainage.

(6) The breaking in of water from the sea or from a river estuary.

(7) The leaving of barriers around small properties or crooked boundaries.

(8) Lack of plans or records showing the extent of old workings, operations of seams not sufficiently proved to justify expenditure for sinking pits; sufficient information might have been obtained if records of previous explorations had been preserved in available form.

(9) The piercing of water-bearing strata by shafts and bore holes which are not protected by water-tight casings, or are not carefully filled and puddled when temporarily left or abandoned.

(10) The cutting through of main faults serving as natural barriers to keep back water and the consequent flooding of the coal.

(11) The leaving of large areas of coal in populous and manufacturing districts to support the surface and the buildings.

While some of the causes mentioned do not apply directly to conditions in this country, the list furnishes a complete synopsis of reasons for coal losses.

Since the issuance of the report of 1871, there have been great improvements in the methods of getting coal. At the present time the long-wall system is in general use, and the waste has been lowered; yet in some parts of the United Kingdom, notably Northumberland, the pillar-and-stall system is still in general use.

Among the factors contributing to a higher rate of recovery is the greatly increased value of small sizes of coal. It was computed in 1871 that the average value of the small coal mined in Great Britain was only 60 cents per ton, while in 1905 the small sizes of steam coal from the South Wales district brought about $1.90 per ton; in all the other coal fields the value has been doubled and even trebled. The principal cause of this change lies in the improved preparation of coal. The manufacture of producer gas on a large scale and the growth of the briquet industry have also increased the possible uses of the small sizes. One of the effects of the increase in the value of small coal has been some decrease of the comparative advantage of the long-wall system, since the production of a large amount of fine coal with the pillar-and-stall system is less objectionable than formerly.*

Interest in the subject continued, and another investigation, more exhaustive than the earlier one, was made by the Royal Commission on Coal Supplies which organized in 1902 and presented its report in 1905. The Royal Commission of 1905 adhered to the limit of depth, namely 4,000 feet, established by the earlier commission. It was thought that, although there might be no insuperable physical or mechanical difficulties in the working of beds at greater depths, the expense would be so great that imported coal could be obtained more cheaply.

With regard to thickness, the commission which reported in 1871 had included seams exceeding one foot in thickness as workable. The question is largely a commercial one, and thinner seams are being worked now than formerly. Mr. Gerrard, inspector of mines for the Manchester district, obtained from all the inspection districts returns which showed that in 1900 17.7 per cent of the entire output was taken from seams not exceeding three feet in thickness.† In the United States, limits of 3,000 feet in depth and of 14 inches in thickness have been decided upon by the Department of the Interior as factors determining what portions of the remaining public lands shall be considered coal lands.‡

The Royal Commission took evidence also on the cost of working, and gave figures which show how greatly the labor cost rises and the individual output declines as thinner beds are mined. Mr. Gerrard gave the underground wages as ranging from $1.68 to $2.28 per ton in seams up to 12 inches, and from 63 cents to $1.36 in all underground seams in his district from 1 foot, 1 inch to 3 feet, while the daily output ranged from one-half ton to 3¼ tons. It was estimated that the cost of digging, loading, and hauling in Scotland was $1.24 for a seam 14 to 15 inches thick, and 65 cents for one from 2 to 2½ feet thick, while the daily output varied from 22 hundredweight to 1½ tons. In Somersetshire the average cost of working thin seams has been about $1.92 per ton for a number of years, while in Yorkshire the cost in 1900 varied from about 96 cents to $1.68.* The Commission of 1905 finally decided to retain the figure of one foot as the limit of thickness.

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*Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXV., 1905.

†Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXXV., 1905.

‡Fisher, Cassius A., "Standards Adopted for Coal Lands of the Public Domain," U. S. Geol. Sur., Bul. 424, Ashley and Fisher, p. 63, 1910.

In connection with the subjects of depth and thickness, it should be noted that it is not the practice in Europe to work single thin beds at great depths. The thin beds are worked in conjunction with thicker ones, and it is the lower cost of production in the latter which makes the working of the thin ones commercially possible. The high cost of working thin beds is partly responsible for the high cost of European coal. The American practice is distinctly different, for there are few, if any, districts in this country in which any bed of bituminous coal is worked unless it is believed that such working shows a profit without reference to other workings. Instances of the working of more than one bed of bituminous coal from the same shaft are rare in the United States.

49. *Percentage of Coal Lost.*—A detailed inquiry was made by the Royal Commission into the various sources of loss. The points of greatest interest in connection with the present study were covered as follows:†

"*Coal left for Support.*—It is evident, that, except in very special cases, it is not possible to remove all the coal. A certain amount must be left in order to maintain shafts, etc., and to support the surface —as, for instance, under houses, railway, canals and rivers—and there seems little hope under existing circumstances of avoiding this source of loss. The amount of coal left for support depends largely upon whether its value is greater than the damage, which would be caused by its removal. . . .

"*Barriers.*—We have evidence that much coal has been and is lost through the practice of leaving unnecessary barriers between royalties and properties; but the present tendency to take large areas under lease is reducing the loss from this cause, and in many cases barriers between properties are now worked out by mutual arrangements.

"*Thick Seams.*—Where the seams are of abnormal thickness much coal is, in some cases, wasted, and for various reasons. Sometimes it is considered that the whole seam cannot be taken out with safety, and part is therefore left to form a roof. Further, such thick seams are more difficult to work, and when the whole of the seam is not of the same quality, there is a temptation to take out the best coal first and to leave the rest for possible future working. Suggestions have been made by some of the witnesses as to the best method of working such thick seams, and there is little doubt that improved methods combined with the increasing use of inferior coal will to a large extent obviate the difficulties mentioned.

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* Digest of Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. 178 et seq., 1905.

† Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXXVI., 1905.

"*Inferior and Small Coal Left in Mines.*—According to the evidence inferior coal is frequently left in the mine owing to its being unsalable, and in some districts considerable quantities of small coal are also left. In recent years there have been vast improvements in the methods of, and the appliances for, preparing and utilizing small and inferior coal, and the higher appreciation of such coal should go far to put an end to this waste."

Table 10 presents the conclusions of the commissioners of different districts regarding the deductions which should be made to cover losses in calculating the amount of coal remaining available.* It is to be understood that the values given do not refer merely to the losses within a definite mined-out area but to the total losses which are to be expected in extracting the total coal remaining available. Since both amounts of losses and reasons for them are governed largely by local conditions, it is unnecessary to go into details, especially since it was found impossible there, as it has been here, to arrive at definite statement for the losses in all cases. Values, however, are founded upon the opinions of men familiar with the practice in the districts, and they are at least approximately correct.

TABLE 10

PERCENTAGES OF COAL LOSSES AS ESTIMATED BY THE ROYAL COMMISSION OF 1905

District	Per Cent lost	District	Per Cent lost	District	Per Cent lost
South Wales and Monmouthshire	20.68	Warwickshire	2.20	Northumberland	22.61
Forest of Dean	15.50	Leicestershire	26.00	Durham	20.23
North Staffordshire	17.00	Lancashire	20.70	Cumberland	28.20
Shropshire	13.00	Cheshire	18.70	Scotland	26.20
South Staffordshire	27.50	Flintshire	27.60		
		Denbighshire	33.30	Average	21.28

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* Ibid, p. XXV.

In the introduction to the report of the Commission, prepared by the editor of the "Colliery Guardian," it is stated (p. xxvi), "Much of the evidence goes to show that the more general adoption of the long-wall system in recent years has resulted in an increased yield of coal. But there are many localities where the conditions are not considered favorable to long-wall working, and where pillars are still left. In the worst cases, in exceptionally bad ground, as much as 50 per cent of the coal is often left behind for this purpose, only to be crushed and oxidized and rendered unfit for future recovery. In the undersea workings in Cumberland as much as 75 per cent is thus left behind. Perhaps the most interesting point brought out in the evidence is that which concerns thick seams. It certainly does seem unfortunate that where there are 9 feet of good coal in a single seam, nearly one-third of this should be left behind. Yet this happens in many of our thickest seams, and the loss threatens to be still more serious as the depth increases."

50. *Mining Conditions on the Continent.*—In the Franco-Belgian basin the beds are for the most part thin, and they are worked, to a considerable extent, at greater depths than those reached in the United States. In Westphalia the beds are mostly steeply dipping, and in Upper Silesia there are combinations of steep dip with great thickness of coal. The development of mining methods in the United States up to the present time has not been affected by practice in these districts.

51. *Percentage of Extraction on the Continent.*—In France it is the custom to extract as much coal as possible from the bed and to fill the resulting space with rock or other material. The filling material is usually transported to its destination in cars, and the method of packing depends largely on the inclination of the bed. In steeply dipping beds the material is allowed to run into place by gravity, but where the slope is not sufficient to permit this method of packing, it is packed by hand. This custom does not entirely prevent subsidence, but it permits the extraction of nearly all the coal without serious disturbance of the surface. While the method of packing followed in these districts permits the removal of nearly all the coal, the removal is accomplished at an expense which would be regarded as prohibitive in the United States in view of the narrow margin between

cost of production and selling price here. The method of filling by flushing is coming into use in France, but has not yet displaced dry filling in most of the mines. Whatever system of filling is used, and whether the coal is taken out by pillar or long-wall method, the extraction is nearly complete.

Some of the most difficult problems found in any coal mining district have been encountered in Belgium. There is no other country in which such thin seams are worked and in which coal is generally mined at such great depths. At Quaregnon a series of thirty-three seams is worked, the average useful thickness being 1 foot, 3½ inches, while the greatest thickness is 2 feet, 2 inches. These beds vary in dip from 8 to 90 degrees. The flatter portions of the bed are worked by long-wall, and the steeper parts by inverted steps forming an interrupted long-wall face. Other beds of nearly the same thickness are being worked, and it appears in all cases that those thin beds are attacked by some form of long-wall working in which, of course, the extraction is practically complete.* The discussion of these districts is much briefer than their importance as coal mining districts would warrant were it not for the fact that the methods used would not in general be adaptable to physical and commercial conditions in this country. They furnish interesting illustrations of high percentages of extraction under difficult conditions, but can hardly be regarded as indicative of what it would be possible to do in the United States.

In the Westphalian district in Germany large amounts of coal have been lost, not so much as the result of poor mining methods or lack of attention to completeness of extraction as because of the necessity of preventing subsidence of the surface. This region is one of great industrial activity, and surface values have so increased within the last half century that high extraction without filling has become impossible. At first, hand filling was employed, the material used being the waste produced in the large amount of rock excavation necessary in beds lying at various angles combined with slack from collieries where coke was not made. More recently the method of hydraulic filling has been introduced. Where the packing is well done and the mining conditions are favorable, the loss of coal is possibly not more than five per cent, which may be considered a fair estimate of the loss even where the long-wall method is followed. There is, however, a greater loss in some of the thicker steeply dipping beds, though it has not been possible to obtain estimates of the amount.

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* Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., pp. 41, 76, 893, 1905.

The Upper Silesian coal field,* situated in the southeast corner of Prussia and extending into Austria and into Russian Poland, has an area of 2,160 square miles. The character of the seam varies considerably both in composition and in thickness, and thick seams occur only in the northern portion of the field where they are very numerous and many of which are of great thickness.

In this coal field, the problem of removing coal beds of great aggregate and individual thicknesses without serious disturbance of the surface has been met by the development of sand flushing processes of filling. This method of filling was borrowed from the anthracite district of the United States where it had first been used.

The mines are worked with and without sand filling. In the method without sand filling much coal is left unworked in the form of pillars and as support under towns or villages. There is a considerable loss resulting from the difficulty of extracting coal left as barriers between the working places and in the old workings. There is also a considerable loss because of fire. The estimated total loss under this method is 25 per cent.

At present sand filling is being used more or less extensively in most of the mines in the thick beds. It is especially advantageous where spontaneous combustion is prevalent and where surface support is necessary. With sand filling when only a part of the coal is replaced by sand it is estimated that the loss of coal is 10 to 15 per cent; with complete replacement of coal by sand filling, the loss is only from 3 to 5 per cent. Smaller and cheaper timber is used in this case, and the greater portion of this timber is recovered for future use. In four mines in Upper Silesia in which sand filling is used extensively and in sufficient quantities to suit the conditions of the mines, the cost in the seams is between 12 and 18 cents per ton. The cost is variable, however, and is calculated in different ways. The average working cost per ton of coal at the surface in this district is $1.51, of which 37 cents is for underground labor.

A report by J. B. Hadesty† shows that the sand filling system has not yet been adopted on a large scale in the western part of Europe, and the statements on cost of filling show that it would be impossible to adopt the process in the United States without materially increasing the cost of production.

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*Gullachsen, Berent Conrad, "The Working of the Thick Coal Seams in Upper Silesia," Trans. Inst. Min. Engrs., Vol. 42, p. 209, 1911.

†"Pennsylvania State Anthracite Mine Cave Commission Report," Journal Pa. Legislature, Appendix, 1913.

It is unnecessary to go into the methods of mining and the results obtained in other coal producing districts. While there are great coal deposits in other parts of the world, and while large quantities of coal are produced, these regions have not been developed sufficiently to work out what may be called a settled practice. No other districts, moreover, are really large producers of coal in the same sense as those already considered. The problems to be considered in connection with districts only partially developed, or districts which though well developed supply only a limited market, are different from those in this country, and the results in such districts are no indication of what can be accomplished here.

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BY

EDWARD C. SCHMIDT, JOHN M. SI
AND
OTTO S. BEYER, Jr.

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ENGINEERING EXPERIMENT STATION

BULLETIN No. 101 **SEPTEMBER, 1917**

COMPARATIVE TESTS OF SIX SIZES OF ILLINOIS COAL ON A MIKADO LOCOMOTIVE

BY
EDWARD C. SCHMIDT
PROFESSOR OF RAILWAY ENGINEERING
JOHN M. SNODGRASS
ASSISTANT PROFESSOR OF RAILWAY MECHANICAL ENGINEERING
AND
OTTO S. BEYER, JR.
FIRST ASSISTANT IN RAILWAY ENGINEERING, ENGINEERING EXPERIMENT STATION

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COMPARATIVE TESTS OF SIX SIZES OF ILLINOIS C ON A MIKADO LOCOMOTIVE

I. INTRODUCTION

1. *Preliminary Statement.*—Until a few years ago practicall of the coal used on locomotives was mine-run coal—the entire screened products of the mines. In recent years, however, increa quantities of screened lump coal have been used in locomotive ser This increase in the consumption of lump coal has been due partl economic factors, such as the increasing market for the screen which result from the production of lump coal; and partly to the b that lump coal, when burned on a locomotive, produces enough steam than mine-run coal to compensate for its greater cost. Sp considerations, such as the desire to lessen the amount of smoke for have also led in some instances to the use of lump coal, which is erally believed to require less skill in firing than mine-run coal. cause of the gradual adoption of mechanical stokers for locomoti the railroads are also using constantly increasing amounts of var sizes of screenings for locomotive fuel. Thus far they have made c paratively little use of any except the sizes mentioned, although tr and market conditions occasionally make it feasible and desirabl employ such sizes as egg, egg-run, and nut coal on locomotives, vided the prices are such as to warrant their use.

Under these circumstances railway purchasing departments continually confronted with the problem of choosing between m run and lump coal, and occasionally with that of choosing betw these and other sizes as well as between various sizes of screeni For such a choice, information regarding the relative values of various sizes of coal in locomotive service is obviously essential; unfortunately very little such information is in existence. Nearly locomotive laboratory tests have been made with mine-run coal, what little information is available concerning the relative values mine-run and lump coal has been derived from road tests, and inadequate and conflicting. There are practically no data concern the other sizes.

An appreciation of the situation thus briefly reviewed, and recognition of the economic importance of reliable information on t subject led, in 1914, to the appointment by The International Railway Fuel Association of a special Committee on Fuel Tests. This committee was instructed to arrange tests in locomotive service for various sizes of coal in order to determine their steam-producing capacities and to define their relative values. This committee held its first meeting in November, 1914, at the University of Illinois, and arrangements then broached in conference between the committee and the representatives of the Engineering Experiment Station of the University later resulted in an agreement for co-operation between the Fuel Association, the University, and the United States Bureau of Mines in carrying on an investigation of the subject under consideration. The tests whose results are here presented constitute the beginning of this investigation. The agreement contemplates the continuation of the research on coals from various other fields. Under the terms of this agreement the University of Illinois has furnished the facilities of its locomotive laboratory, the services of the staff of its department of railway engineering, and a portion of the funds required for the tests; the Fuel Association has provided the remainder of the funds; and the Bureau of Mines has made all the chemical analyses and the heat determinations of the coal, ash, and cinders. In perfecting these arrangements, the Fuel Association was represented by the committee whose members are named in Section 2; and the Bureau of Mines, by Director Van. H. Manning, and Mr. O. P. Hood, Chief Mechanical Engineer.

The funds supplied by the Fuel Association were obtained by subscription and did not become available until the Spring of 1916; the locomotive, then under construction, was not delivered until the Fall of that year. The tests were begun in December, 1916, and were completed in February, 1917.

The body of this bulletin contains information concerning the test program, the coal, the locomotive, the laboratory, the test methods and conditions, and the results. Appendixes I, II, and III contain more detailed statements regarding the locomotive and the methods, and complete tabulated results. In Appendix IV, there are presented certain data relating to engine performance. In Appendix V, there are the results of a few of the tests which, in order to study the uniformity of conditions during their progress, were divided into three periods, and the data for each period were separately calculated.

The results of this investigation have already been presented in a report to the International Railway Fuel Association Convention held in Chicago in May, 1917; and, in somewhat different form, will appear in the Proceedings of the Association for this year.

2. *Acknowledgments.*—The Committee on Fuel Tests previously referred to, under whose direction the work was planned and the general program defined, was composed of the following:

J. G. CRAWFORD, Fuel Engineer, Chicago, Burlington & Quincy Railroad, *Chairman*

H. B. BROWN, General Fuel Inspector, Illinois Central Railroad

W. P. HAWKINS, Fuel Agent, Missouri Pacific Railway System

O. P. HOOD, Chief Mechanical Engineer, United States Bureau of Mines

L. R. PYLE, Fuel Supervisor, Minneapolis, St. Paul & Sault Ste. Marie Railroad

W. L. ROBINSON, Supervisor of Fuel Consumption, Baltimore & Ohio Railroad

E. C. SCHMIDT, Professor of Railway Engineering, University of Illinois

The locomotive used during the tests was loaned by the Baltimore and Ohio Railroad Company, through the interest and courtesy of Mr. J. M. DAVIS, Vice President; Mr. F. H. CLARK, General Superintendent of Motive Power; and Mr. M. K. BARNUM, Assistant to the Vice President. The tests came at a time when traffic demands were extraordinary, and the loan of the locomotive constituted as great a contribution to the work as that made by any other agency.

The funds provided by the International Railway Fuel Association were donated to the Association by the following railroads, coal companies, and railway supply manufacturers:

ATCHISON, TOPEKA, AND SANTA FE RAILWAY
ATLANTIC COAST LINE RAILWAY
BALTIMORE AND OHIO RAILROAD
CHICAGO GREAT WESTERN RAILWAY
CHICAGO, INDIANAPOLIS, AND LOUISVILLE RAILWAY
ERIE RAILROAD
LONG ISLAND RAILROAD
MINNEAPOLIS, ST. PAUL, AND SAULT STE. MARIE RAILWAY
NORFOLK AND WESTERN RAILWAY

St. Louis Southwestern Railway
Seaboard Air Line Railway
Big Muddy Coal and Iron Company
T. C. Keller and Company
Old Ben Coal Corporation
W. P. Rend and Company
Southern Coal and Mining Company
Taylor Coal Company
United Coal Mining Company
American Arch Company
American Locomotive Company
Franklin Railway Supply Company
Locomotive Stoker Company
Locomotive Superheater Company
The Pilliod Company

The Locomotive Stoker Company and the Baltimore and Ohio, the Chicago and Northwestern, the Erie, and the Minneapolis, St. Paul & Sault Ste. Marie Railroad Companies each delegated to the laboratory a man to act as test observer and calculator for the entire period of the tests. Mr. L. R. Pyle, Fuel Supervisor of the road last named, was in charge of the cab operations and supervised the work of the fireman. The uniformity attained in the firing and in the conditions of combustion was due largely to the experience and skill of Mr. Pyle.

The department of mining engineering of the University of Illinois contributed the use of its laboratory facilities for crushing and sampling the coal and analysing the flue gas; and Professors H. H. Stoek and E. A. Holbrook of that department gave advice on many matters connected with the investigation. The laboratory coal screen used in the tests was designed by Professor Holbrook.

II. Purpose and Program

As has been stated, the ultimate purpose of the tests was to determine the relative values of different sizes of coal when burned on a locomotive. The immediate purpose was to find for each size, at two rates of evaporation, the number of pounds of water evaporated per pound of coal, in the expectation that these values of evaporation would provide a proper basis for comparing the performance of the sizes and for defining their relative values. The tests were made on a Mikado (2-8-2) type locomotive.

Six sizes of Franklin County, Illinois, coal were selected- run, 2-inch by 3-inch nut, 3-inch by 6-inch egg, 2-inch lump, screenings, and 1¼-inch screenings. The general test progr volved for each size of coal six tests, three of which were mad medium rate of evaporation, and the remaining three at a hig The medium rate was chosen to represent an average rate of w the locomotive, in so far as it is possible to define such an a During tests run at this medium rate about 23,000 pounds of were evaporated per hour under the prevailing conditions of pressure, superheat temperature and feedwater temperature; 3,100 to 4,300 pounds of coal were fired per hour; and the engi worked at 33 per cent cut-off and at about 19 miles per hour, d ing approximately 1,300 indicated horse power and about pounds drawbar pull. During tests when the engine was wor the high rate of evaporation, about 43,000 pounds of water were rated per hour, the hourly coal consumption varied from abou to 9,300 pounds, the cut-off and speed were respectively 55 pe and 26 miles per hour, while the horse power was about 2,200, a drawbar pull about 28,500 pounds.

The number of tests actually run with each size at each evaporation was as follows:

Size of Coal	No. of Tests at the Medium Rate of Evaporation	No. of T the High of Evapo
Mine Run	3	3
2" x 3" Nut	4	3
3" x 6" Egg	3	3
2" Lump	3	4
2" Screenings	3	2
1¼" Screenings	3	2

III. The Coal Used

3. *Source and Mining Methods.*—All coal used during t was secured from the United Coal Mining Company's Mine located one mile east of Christopher, Franklin County, Illin the Illinois Central and the Chicago, Burlington and Quinc roads. This mine was chosen by the Fuel Association Commi cause western railroads draw a large fuel supply from this fie because of its nearness to the locomotive laboratory.

The coal is derived from what is designated by the Illinois cal Survey as bed No. 6 of the Carboniferous Age, Carbondale formation. The bed averages in thickness at this mine about 9 feet and carries almost throughout, at from 18 to 30 inches from the floor, a "blue band" variable in thickness and consisting of "bone," shaly coal, or gray shale. The mine is worked under the room-and-pillar system, and the coal is undercut with electric chain machines. It separates at a parting of mother coal about 14 to 30 inches from the top of the bed, and the coal above this parting is left for the roof. The coal face and the mine itself are quite uniformly dry.

All the coal was mined, screened, and loaded by the methods usually employed at the mine for supplying the ordinary commercial product. It was inspected during the process of loading by one of the regular fuel inspectors of the Chicago, Burlington and Quincy Railroad, who at the time of inspection took at the tipple samples for analysis, the results of which were later used to compare the moisture in the coal when loaded with its moisture content when used at the laboratory.

While it was originally planned to ship all test coal in box cars to protect it from the weather during transit and before it could be unloaded at the laboratory, only two cars of mine-run coal were so shipped. Under the prevailing conditions of business and car supply, the plan proved impracticable and had to be abandoned, and all coal except these two car loads was shipped in ordinary flat-bottomed gondola cars. As promptly as possible after its receipt at the laboratory—on the average 6 days, and in no instance more than 12 days after its arrival—the coal was unloaded into covered bins where it remained protected from the weather until used. The cars were unloaded by hand shoveling about as they would have been at some of the older types of railway coal pockets, and the coal was probably subjected to about the same amount of breakage in this process. The maximum time which elapsed between loading the coal at the mine and testing it was 37 days in one instance. Taking the tests as a whole the average time between loading and testing was about 25 days.

4. *Preparation.*—At the mine all coal was dumped from the mine cars into a hopper from which it was run out on a stationary deadplate where it spread out before reaching the shaking screens. The various sizes were prepared as follows:

The mine run coal was the entire unselected product of the mine.

The 2-inch lump was made by passing mine run coal over a screen having 144 square feet area with 2-inch round openings, and consisted of everything going over this screen.

The 2-inch by 3-inch nut consisted of what passed over the previously mentioned screen and through a screen having 72 square feet area with 3-inch perforations; and it was re-screened over a stationary screen having an area of 18 square feet with slots ⅞-inch wide and 8-inches long, and a stationary screen of 20 square feet area with 1¼-inch round perforations.

The 3-inch by 6-inch egg passed first over a screen having 144 square feet area with 2-inch round perforations and 72 square feet area with 3-inch round perforations, and then through a screen having 32 square feet area with 6-inch round perforations.

The 2-inch screenings were passed through the screen over which the 2-inch lump coal was made, namely, 144 square feet area with 2-inch round perforations.

The 1¼-inch screenings were made at the re-screening plant through a revolving screen having 411 square feet area of plate with ¾-inch round perforations and 188 square feet area of plate with 1¼-inch perforations.

5. *Chemical Analyses.*—During the progress of each test, while the coal was being loaded into the charging wagons to be taken to the firing platform, samples were taken for the purpose of analysis. These samples varied in amount from 500 to 1000 pounds, and they were taken according to methods prescribed by the American Society for Testing Materials as set forth in the year book of the society for 1915. The sampling process is described in Appendix II. Under arrangements made with the United States Bureau of Mines, all analyses of coal, ash, and cinders were made at the laboratories of the bureau in Pittsburgh, where the samples were shipped immediately upon the conclusion of each test.

The results of these analyses are given for each test in the tables in Appendix III. The averages of the coal analyses for all tests made with each grade of coal are presented in Table I. An inspection of this table reveals a rather unusual uniformity among the various sizes with regard to their composition and heating value. Considering all six sizes, the ash content varied from 8.06 per cent to 10.59 per cent and the heating value per pound of dry coal varied from 12,711 to 13,239 B. t. u. The analyses for the two sizes of screenings correspond very closely in all respects and their average heating value

TABLE 1

THE CHEMICAL ANALYSES AND HEATING VALUES OF THE COALS

(The table gives the averages for all tests for each size.)

SIZE OF COAL	Proximate Analyses—Coal as Fired					Calorific Values			Ultimate Analyses—Coal as Fired				Moisture in Coal Determined from Sample taken at Mine per cent
	Moisture per cent	Volatile Matter per cent	Fixed Carbon per cent	Ash per cent	Sulphur Separately Determined per cent	Per lb. of Coal as Fired B. T. U.	Per lb. of Dry Coal B. T. U.	Per lb. of Combustible B. T. U.	Carbon per cent	Hydrogen per cent	Nitrogen per cent	Oxygen per cent	
Mine Run.........	8.14	34.18	47.92	9.76	0.95	11873	12926	14463	66.63	4.28	1.55	8.69	7.82
2″ x 3″ Nut........	8.60	34.83	47.70	8.87	0.88	11957	13082	14487	67.50	4.36	1.38	8.42	8.48
3″ x 6″ Egg........	8.82	34.57	48.56	8.06	0.94	12071	13239	14523	68.19	4.50	1.51	7.99	.....
2″ Lump..........	9.27	34.46	47.49	9.07	0.88	11817	13023	14469	66.34	4.23	1.49	8.73	.....
2″ Screenings......	9.25	32.05	48.12	10.59	0.85	11550	12727	14408	65.74	4.43	1.48	7.66	.....
1¼″ Screenings....	9.09	32.34	48.01	10.57	0.97	11557	12711	14385	65.49	4.35	1.43	8.10	9.07

based on dry coal, was only about two per cent less than the average heating value of the four large sizes. Their average ash content was 10.58 per cent, and the average ash for the other sizes was 8.94 per cent—a difference of 1.64 per cent. As would be expected, the mine run occupies an intermediate position between the screenings and the egg, nut and lump, both with regard to ash content and to heating value. The uniformity of the analyses and of the heating values makes it clear that such differences in performance as developed between the various sizes are due chiefly to differences in their mechanical make-up, and only in small measure to differences in their chemical composition. This fact is further emphasized by discussion which appears later in the report.

6. *The Make-up of the Coals as Received.*—Because of differences in the nature of the coal, in mining methods, and in methods of preparation, there is frequently much uncertainty about the meaning of such terms as " mine run," " lump," etc. The mine run grade from a district where the coal is soft and friable, for example, is likely to contain a larger proportion of fine coal than mine run made from a harder coal. Similarly the methods of mining, the use of bar instead of plate screens, or square-hole instead of round-hole screens, all entail differences in the make-up of coals which are designated by identical names. For these reasons the laboratory has devised a method of screening samples of the coals used during tests for the

purpose of separating them into their size elements in orde able to define and record the actual mechanical make-up of the grades.* All the coals used in these tests were thus screen this screening process is referred to in the report as the mec analysis.

The samples for this purpose were taken while the cars wer unloaded, by methods which are described in Appendix II. carloads each of mine run and lump, and two carloads of each other four grades were received at the laboratory. For both t run and the lump coals, two of the three carloads of each si sampled for screening. Samples were taken from each car of each car of egg, whereas the two cars of 2-inch screenings and cars of 1¼-inch screenings were combined for each size, and o ple only was taken from each. There was thus taken for mec analysis a total of ten samples, each of which weighed about t

These samples were screened by means of the specially d shaker screen shown in Fig. 1. This consists of two incline frames each of which is supported by four vertical wood springs. These frames are shaken by connecting rods atta the pulley-driven eccentrics which appear at the right of the and which were run at a speed of 80 revolutions per minut frames carry removable plate screens provided with round ations. Five such screens were used perforated respectivel 4-inch, 2-inch, 1-inch, ½-inch, and ¼-inch holes.

Starting with the 4-inch screen in the upper and the 2-inch in the lower frame, one of the samples—mine run, for exa was fed over the upper frame and the coal was separated i parts; one containing what passed over the 4-inch screen, th what passed through the 4-inch screen and over the 2-inch scre the screenings which passed through the 2-inch screen. The fi portions were then set aside for weighing, the screens were repl the plates with 1-inch and ½-inch holes, the screenings wer fed onto the upper plate and the process repeated, ending final the ¼-inch screen. In this way the sample was divided into si whose size limits were as designated by the headings of Col to 7 in Table 2. These parts were then weighed and the ratios weights to that of the original sample were calculated.

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*The term "grade" is occasionally used throughout this bulletin instead of "size." It refers solely to one of the six sizes tested, and does not imply any dif quality or kind.

Table 2 presents the average values of these ratios and it defines, therefore, for each grade the magnitude of the size elements which went to make up the original coal and thus records definitely its composition. The significance of Table 2 is perhaps made clearer by Figs 3 to 8 inclusive. Each of these illustrations applies to one of the sizes and each figure is reproduced from a photograph of the various size elements which came from the screen and which, after weighing, were assembled side by side as shown in the cuts. These figures present graphically the same information as is given in Table 2. Fig. 2 is reproduced from photographs of the original coal samples and represents the six sizes as they were received at the laboratory.

TABLE 2

THE SIZE ELEMENTS OF THE COALS AS RECEIVED AT THE LABORATORY

(This table gives the direct results of the separation made by the use of the laboratory screens)

SIZE OF COAL	Per Cent over 4" Screen	Per Cent through 4" over 2" Screen	Per Cent through 2" over 1" Screen	Per Cent through 1" over ½" Screen	Per Cent through ½" over ¼" Screen	Per Cent through ¼ "Screen	Total
1	2	3	4	5	6	7	8
Mine Run........	29.6	22.3[1]	16.8[1]	11.4	7.4	12.5	100.0
2" x 3" Nut	....	63.9	30.3	2.8	1.1	1.9	100.0
3" x 6" Egg	41.0	48.3	5.3	2.0	1.1	2.3	100.0
2" Lump..........	61.6	26.4	7.5	1.9	.9	1.7	100.0
2" Screenings......	....	....	33.2	25.7	14.2	26.9	100.0
1¼" Screenings ...	....	....	4.5	37.9	20.0	37.6	100.0

[1] Derived from plotted curves (Fig. 9).

The facts presented in Table 2 may be re-combined to permit tabular and graphical definitions of the grades in another form. Considering in Table 2 the 2-inch by 3-inch nut coal, if we add Columns 4 to 7 we find that 36.1 per cent of this coal passes through a 2-inch screen. Adding Columns 5, 6, and 7 we find that 5.8 per cent will pass through a 1-inch screen, etc. Obviously also 100 per cent of this grade passed a 3-inch screen in the original preparation at the mine. The total per cents passing the various sized screens determined in this manner from Table 2 are assembled in Table 3, where we find that for the 2-inch by 3-inch nut coal, 31.1 per cent, 5.8 per cent, 3.0 per cent, and 1.9 per cent passed respectively 2-inch, 1-inch, ½-inch, and ¼-inch screens. If now we plot as in Fig. 9 the percentages given in Table 3, together with the corresponding screen size, we get for the nut coal curve No. 3 there drawn, which serves to

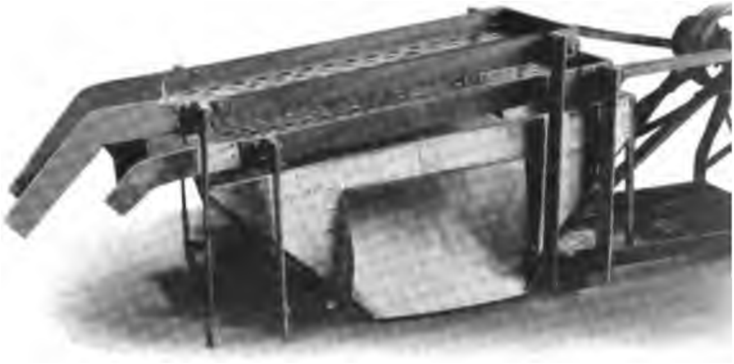
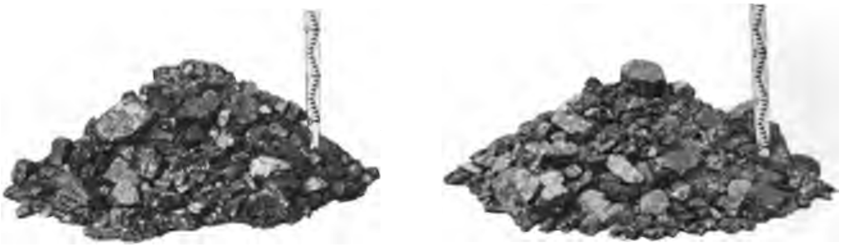
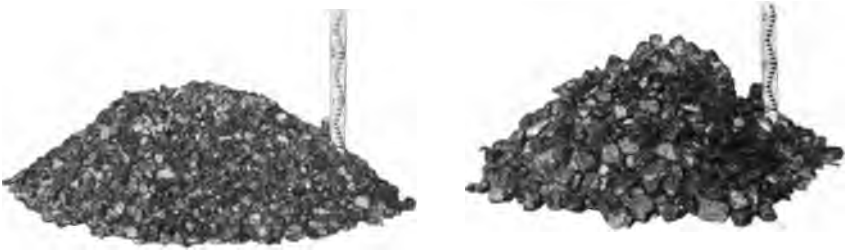


FIG. 1. THE LABORATORY COAL SCREEN



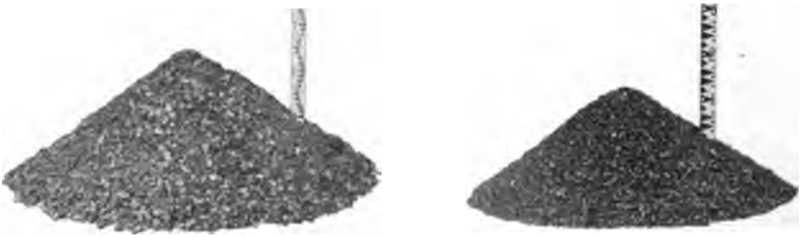
MINE RUN

2" LUMP



2"X3" NUT

3"X6" EGG



2" SCREENINGS

1¼" SCREENINGS

FIG. 2. THE SIX SIZES OF COAL USED DURING THE TESTS, IN THE CONDITION IN WHICH THEY WERE DELIVERED AT THE LABORATORY

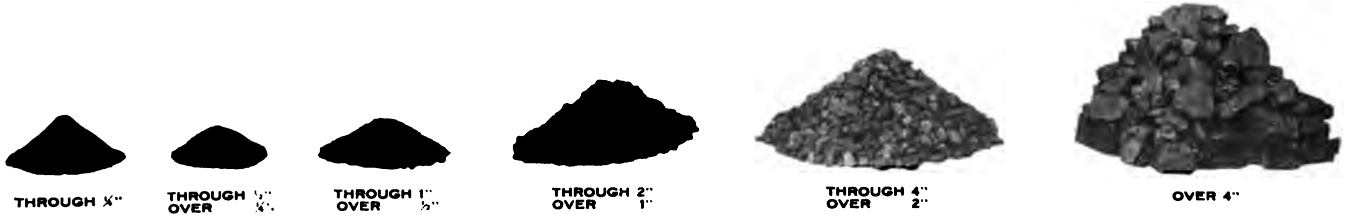


Fig. 3. The Size Elements of the Mine Run Coal

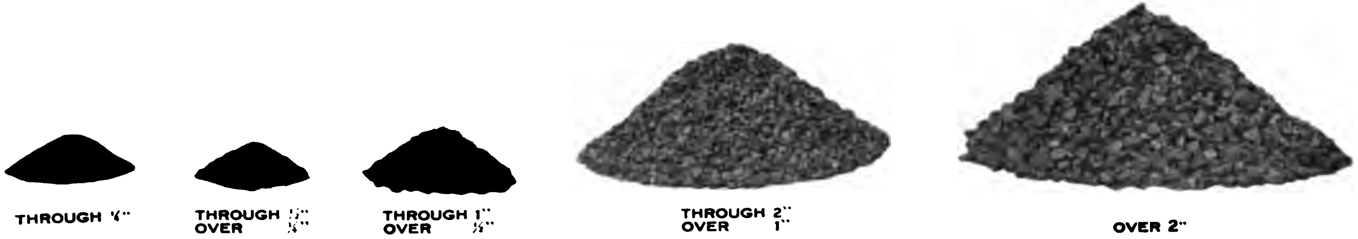


Fig. 4. The Size Elements of the 2-inch by 3-inch Nut Coal

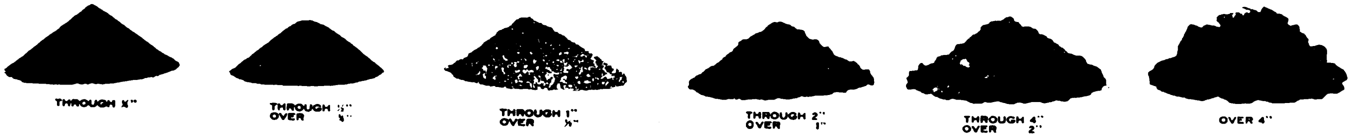


Fig. 6. The Size Elements of the 2-inch Lump Coal

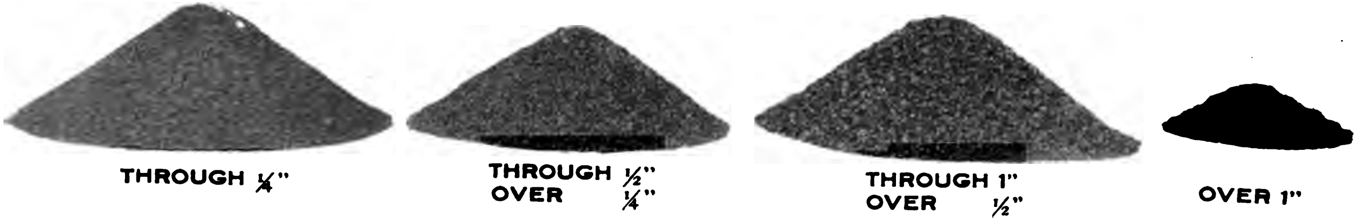


Fig. 7. The Size Elements of the 2-inch Screenings

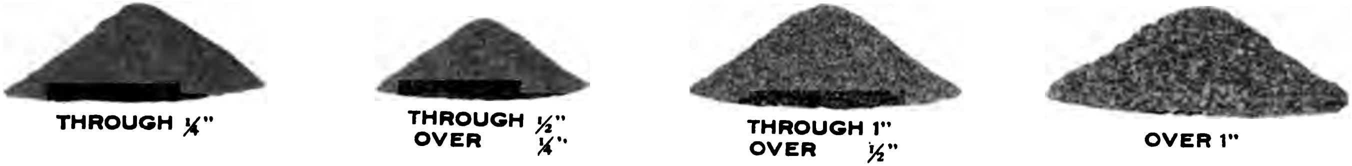


Fig. 8. The Size Elements of the 1¼-inch Screenings

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The titles to Fig. 3 and Fig. 6 should be interchanged, and

The titles to Fig. 7 and Fig. 8 should be interchanged.

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s through th

marked in Fig. 9, but presumably to determine th screens of any intermediate size. The six curves d plotted from the percentage values and the scr Table 3 for each of the grades. Those portions o with broken lines are not supported by direct expe scale shown in the upper part of the diagram repres which are commonly used in the mines of southern

TABLE 3

THE MAKE-UP OF THE COALS AS RECEIVED AT TH

(This table presents the results computed from

SIZE OF COAL	Per Cent over 4" Screen	Per Cent through 4" Screen	Per Cent through 2" Screen	Per Cent through 1" Screen
1	2	3	4	5
Mine Run......	29.6	70.4	48.1	31.3
2" x 3" Nut.....	....	....	36.1	5.8
3" x 6" Egg.....	41.0	59.0	10.7	5.4
2" Lump.......	61.6	38.4	12.0	4.5
2" Screenings ...	....	....	....	66.8
1¼" Screenings .	....	....	....	95.5

If in Fig. 9, we follow curve No. 5 pertainin we find that about 90 per cent of it will pass thr 9-inch round holes; about 87 per cent of it will pa screen; about 70 per cent through a 4-inch screen; a 2-inch screen, and so on. It is interesting to not 3-inch nut, the 3-inch by 6-inch egg, and the 2- nearly the same proportions of coal which passes t or less in diameter; whereas in these sizes the pro coal differ materially. Other comparisons are re having all six sizes thus represented on one diag borne in mind that the curves in Fig. 9 define the the condition in which they were unloaded fro laboratory.

7. *The Make-up of the Coals as Fired.*—All mine run and lump were unloaded into the cha the bins without further preparation, and they fired in exactly the condition in which they arrive

except for the breakage incident to unloading and the insignificant breakage due to shoveling into the charging wagons.

Since, however, the mine run and the lump coals contained as usual a considerable proportion of lumps too large for proper firing, the attempt was made to break these two sizes down to the extent to which, in the judgment of those in charge of the tests, these grades are generally broken down at the coal chute. These two coals as fired contain, therefore, a smaller proportion of large lumps than when they were received and the extent to which this extra preparation modified the make-up of the coals is defined in the table, the figures, and the discussion which follow.

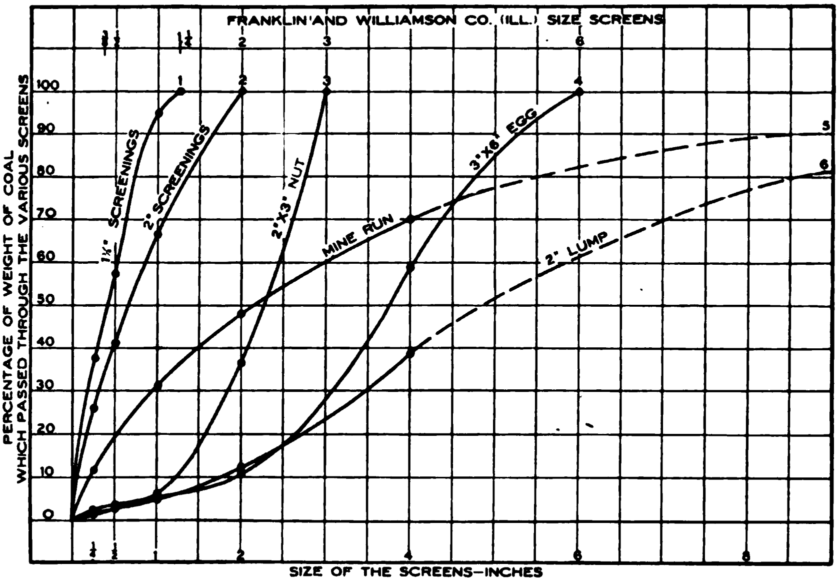


FIG. 9. THE MAKE-UP OF THE COALS IN THE CONDITION IN WHICH THEY WERE RECEIVED

After the mechanical analysis samples taken to represent the mine run and lump coals as received at the laboratory had been screened and separated as described in the preceding section, the large lumps in each sample were broken down to the same extent as these sizes were broken before firing, and under the supervision of the same test operators who controlled this process during the progress of the tests.

These reduced samples for mine run and lump coal were accepted representing these two grades in the condition in which they were fir and they were subjected to the same screening process as has be previously described. The results of this mechanical analysis are p sented for these two coals "as fired" in Table 4, and the size p centages of these grades in the condition in which they were receiv are also embodied for comparison's sake in the same table.

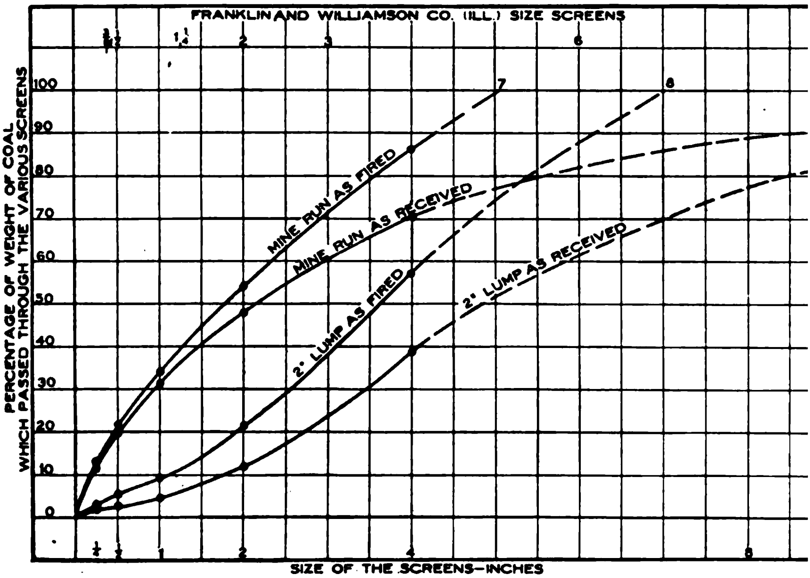


FIG. 10. THE MAKE-UP OF THE MINE RUN AND THE LUMP COALS, AS RECEIVED AND AS FIRED

The values in Table 4 are plotted in Fig. 10, in which curve No applies to mine run coal as received and No. 7 to the same grade fired; while curves 6 and 8 apply to the 2-inch lump coal in the co ditions as received and as fired, respectively. The extent to whi these two sizes were broken down is revealed by an inspection Table 4 and Fig. 10. Considering the curves in the figure, it is a parent that the largest lumps in the mine run were somewhat furth broken down than those in the 2-inch Lump. After reduction mine run passed through a 5-inch screen, whereas only about 74 p cent of the lump would pass a screen of this size.

TABLE 4

THE MAKE-UP OF THE MINE RUN AND THE LUMP COAL, BOTH AS RECEIVED AND AS FIRED

SIZE OF COAL	Per Cent over 4″ Screen	Per Cent through 4″ Screen	Per Cent through 2″ Screen	Per Cent through 1″ Screen	Per Cent through ½″ Screen	Per Cent through ¼″ Screen
1	2	3	4	5	6	7
Mine Run:						
As Received..	29.6	70.4	48.1[1]	31.3	19.9	12.5
As Fired.....	13.1	86.9	54.3[1]	34.3	22.0	13.8
2″ Lump:						
As Received..	61.6	38.4	12.0	4.5	2.6	1.7
As Fired......	42.7	57.3	21.0	9.0	5.3	2.0

[1] Derived from plotted curves (Fig. 10).

In order to permit comparisons of the mechanical make-up of all six sizes as fired, curves Nos. 1, 2, 3, and 4 from Fig. 9 (applying to the grades which were fired as received) and curves 7 and 8 from Fig. 10 are brought together in Fig. 11, which consequently shows the make-up of all the grades in the condition in which they were fired during the tests.

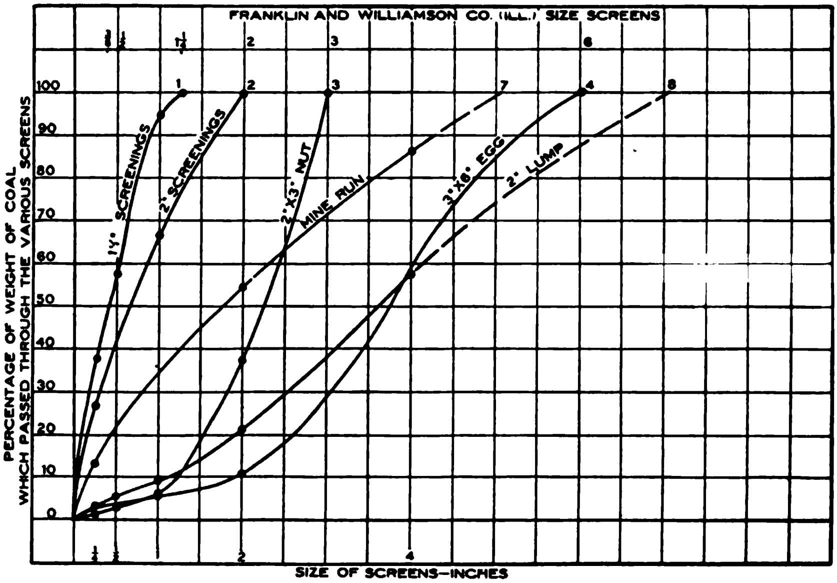


FIG. 11. THE MAKE-UP OF THE COALS, IN THE CONDITION IN WHICH THEY WERE FIRED

IV. THE LOCOMOTIVE

8. *Design and Main Dimensions.*—The locomotive used during the tests was loaned for the purpose by the Baltimore and Ohio Railroad Company. It was of the Mikado type (2-8-2); its road number was 4846, and its classification, Q-7-F. It was built by the Baldwin Locomotive Works during the summer of 1916, entered service in September, and upon its arrival at the laboratory, had run approximately 3400 miles. It arrived at the laboratory in excellent condition.

The principal dimensions of the locomotive are as follows:

Total weight, in working orders, lb.	284,500
Weight on drivers, lb.	222,000
Cylinders (simple), diameter and stroke, in.	26 x 32
Diameter of drivers, in.	64
Firebox, length and width, in.	120 x 84
Firebox volume, cu. ft.	348.6
Grate area, sq. ft.	69.8
Heating surface, 2¼-inch tubes (fire side), sq. ft.	2410
Heating surface, 5½-inch tubes (fire side), sq. ft.	973
Heating surface, firebox and tube sheets (fire side) sq. ft.	247
Heating surface, total (fire side) sq. ft.	3630
Heating surface, superheater (fire side) sq. ft.	1030
Boiler pressure, lb. per sq. in.	190
Tractive effort, lb.	54587

The boiler was of the wagon-top type with radial stays. It was equipped with a Schmidt top-header superheater consisting of 34 elements, a Street stoker, and a Security brick arch carried on four tubes. The front end was self-cleaning and was equipped with a plain 6-inch nozzle-tip without bridge or split, which was used throughout all tests.

The grates were of the box type, the design of which is shown in detail in Appendix I. The total air opening through the grates amounted to 17 square feet or 24.4 per cent of the grate area. The area of the air inlet to the ash pan amounted to 8.3 square feet or 49 per cent of the air opening through the grates.

The locomotive was regularly equipped with a hand-operated door which was replaced, however, during the period of the tests by a Franklin pneumatic door of the butterfly type. This was used during all tests except those with the two sizes of screenings, which were fired by means of the Street stoker.

The design of the locomotive is described in further detail in Appendix I.

9. *Inspection.*—In order to ensure uniformity of its condition as regards accumulations of scale, soot, etc., and to define these conditions, the locomotive, during its stay in the laboratory, was handled and inspected as follows:

The boiler water was changed every five or six days and the boiler was washed about every two weeks. More frequent washings were unnecessary because the laboratory water is relatively free from scale-forming salts and suspended matter. Monthly inspections in accordance with Interstate Commerce Commission regulations were made during the test period as during regular service.

During the progress of the earlier tests, in order to ensure uniformity in the condition of the heating surfaces, all tubes, flues, and superheater elements were blown free from soot and small "honeycomb" immediately before each alternate test. Although there was nothing in the test results to indicate that this was not being done often enough, to remove all uncertainty this blowing down process was gone through before each test during the latter part of the series. The front end netting was cleaned of all cinders, and the cinders were removed from the top of the arch at the same time. The arch was inspected after each test, and all defective bricks were immediately replaced.

Upon its arrival at the laboratory, the interior of the boiler was inspected and was found to be free from all scale except a very thin coating. The laboratory water itself is not only nearly free from scale-forming materials but it tends frequently to soften hard scale deposited by other water; and a final interior inspection revealed the fact that this softening had occurred in this instance, and that the original scale had been considerably disintegrated. There was nothing, however, in the change in the scale to indicate that the heat transfer through the surfaces had, in any significant degree, varied on account of scale during the progress of the tests. The facts that the boiler had run only 3400 miles before arriving at the laboratory and that its mileage during the tests was only 3600, are perhaps in themselves sufficient evidence that there could have been no accumulation of scale nor change in its thickness sufficient materially to affect the test results.

V. THE LABORATORY

The locomotive laboratory is fully described in Bulletin 82 of the Engineering Experiment Station of the University of Illinois; descriptions have been published also in the Proceedings of the American

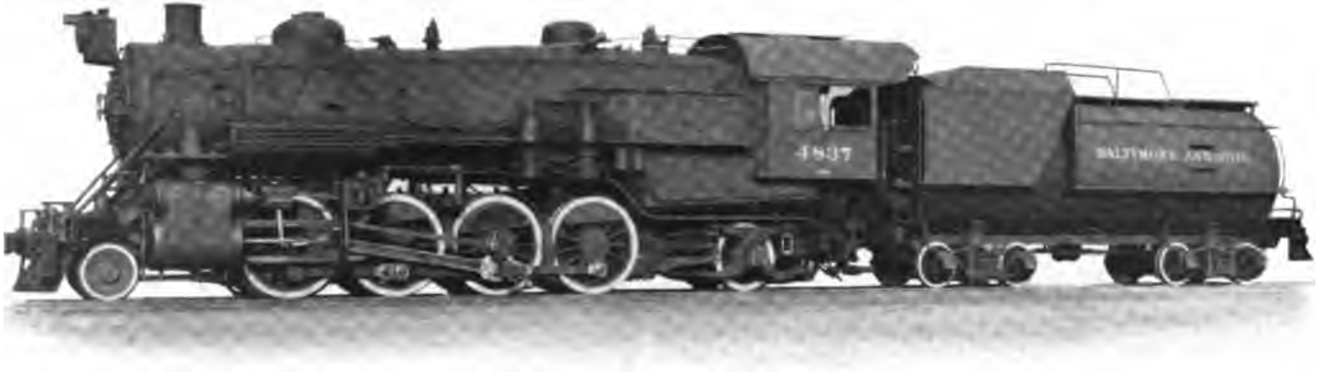


FIG. 12. BALTIMORE AND OHIO RAILROAD LOCOMOTIVE, 4837, IDENTICAL IN DESIGN WITH THE ONE USED IN THE TESTS

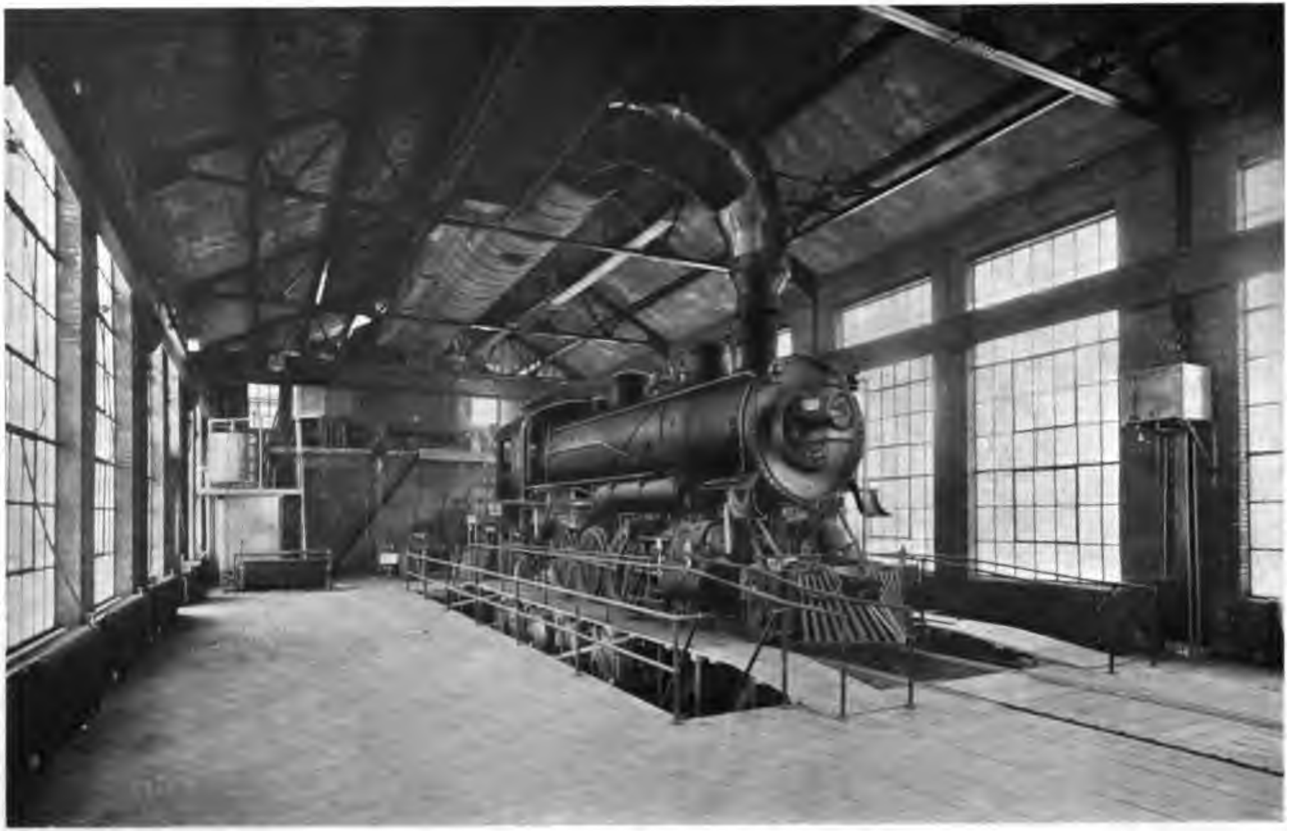


Fig. 13. An Interior View of the Laboratory, with a Locomotive in Test Position.

Railway Master Mechanics' Association, Vol. 46, 1913, and in the Proceedings of the Western Railway Club for March, 1913. It is unneccessary, therefore, to include here any detailed statement concerning its design. Since, however, the amount of the cinder losses in this series of tests serves in large measure to account for the differences in performance of the various grades of coal, these losses have an especial significance; and it seems appropriate therefore to describe briefly the cinder separator by means of which they were determined.

All gases and exhaust steam are discharged across an open space above the locomotive stack into the mouth of a large steel elbow which carries them up and over to a horizontal duct running through the roof trusses to the rear of the building, terminating at an exhaust fan. This elbow and duct are illustrated in Fig. 13. The gases and steam are drawn through the duct by the fan and they are then passed through a breeching to the cinder separator itself which is located outside the building and forms the base of a stack through which the gases are finally discharged to the air. The separator and stack are shown in section in Fig. 14.

The cinder-laden gases enter the separator at *B* and, in order to leave, they must pass downward and around the sleeve *A*. This passage gives them a whirling motion, which causes the cinders by centrifugal force to move toward the outside wall along which they fall to the hopper below, while the gases pass out through the sleeve to the stack. The cinders collecting at the bottom of the hopper are drawn off, weighed, and analysed. This separator collects all solid matter which issues from the locomotive stack, except possibly the finest dust. The cinders taken from the separator during tests with mine-run coal have contained from 10 to 18 per cent of material which passed a screen with 200 meshes to the inch.

VI. FIRING METHODS

It is desired to present at this point only such information concerning the methods used in the laboratory as relates to the measurements of coal and to the firing. Information about methods relating to other test processes is given in Appendix II.

10. *Coal Measurements.*—Before starting the test, the engine was run at the desired load and speed long enough beforehand to permit the rate of feed through the injectors to be adjusted to the test conditions, and it was generally unnecessary to change this rate

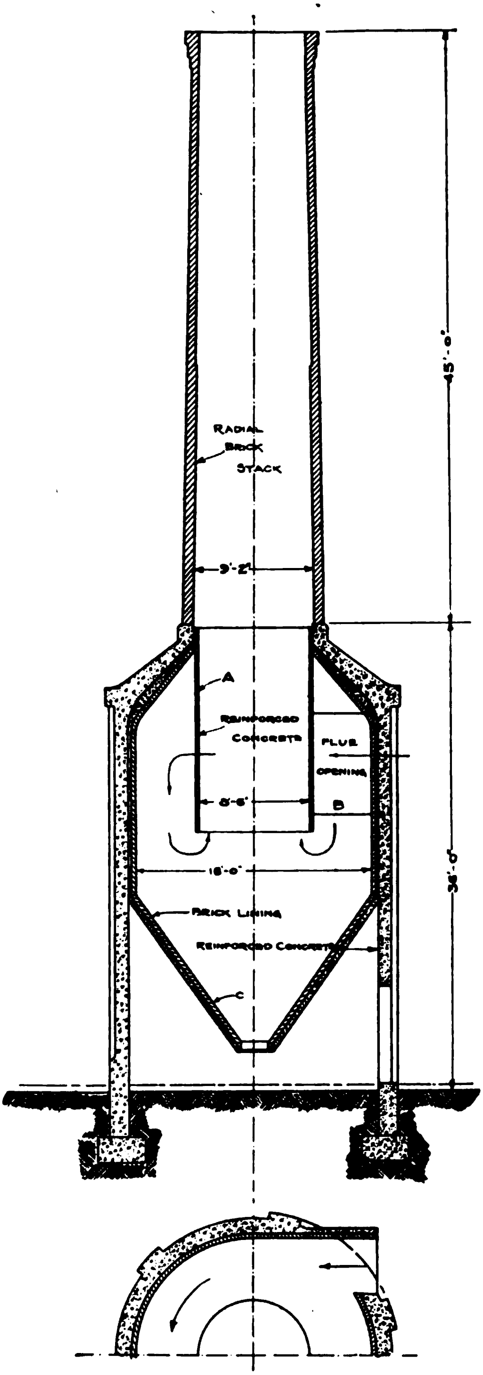


FIG. 14. CROSS SECTION OF THE CINDER COLLECTOR AND STACK.

during the progress of the test. The tests were not started until the desired conditions of load, speed, boiler pressure, and a proper condition of the fire had been established. The coal was weighed in one-thousand pound lots before being delivered to the firing platform. Throughout each medium rate test, the time of firing the last scoopful of each ton was recorded, together with the levels of the water in the main feed tank and in the boiler gauge glass. During the high rate tests, these facts were recorded at the time of firing the last scoopful of each two tons of coal. This procedure made it possible to control the regularity of the firing process and it also makes available facts which may be used to illustrate the regularity of feed of both the coal and the water. For this purpose tests 2405 and 2416, fairly characteristic of the series, have been chosen. During test 2416, run at a medium rate of evaporation, the time required to fire each of the ten successive lots of 2000 pounds, varied only from 34 to 36 minutes; and the amount of water fed per minute during these ten intervals varied only from 390 to 413 pounds. During test 2405, which was run at a high rate of evaporation, the times required to burn each of the five successive lots of 4000 pounds of coal were respectively 36, 33, 31, 32 and 31 minutes; and the water fed per minute during these intervals varied only from 693 pounds to 709 pounds.

11. *Firing Methods.*—The locomotive was fired throughout all tests by Mr. C. Welker, a skilled fireman, who was detailed by the Illinois Central Railroad from its regular force. Previous to the tests he had had about seven and one-half years' experience firing in service, and he had also had about a half year's experience as fireman in the laboratory. The supervision of the fireman, the control of the injectors, and other cab operations were, during all except the last three tests, in charge of Mr. L. R. Pyle, Fuel Supervisor of the Minneapolis, St. Paul and Sault Ste. Marie Railroad, and member of the Fuel Test Committee. During the last three tests Mr. Pyle's place was taken by Mr. B. F. Crolley, Supervisor of Locomotive Operation of the Baltimore and Ohio Railroad. During the tests of the two sizes of screenings, which were fired by the stoker, the firing was supervised by Mr. E. Prouty, Mechanical Expert of the Locomotive Stoker Company.

All hand-fired tests were fired by the "three-scoop system," that is, three scoopfuls of coal were fired at a time, both during the medium and the high rate tests, although during the latter it was necessary to cut down the interval between firings. The fireman was instructed to keep these intervals as regular as possible and in this he was aided from time to time by stop watch observations. The degree of regularity attained is evidenced by the figures concerning the rate of coal consumption which have already been cited, and by the graphical logs shown in Appendix III. During the stoker-fired tests, the same regularity of feed was sought and attained. No coal was fed by hand during any of these tests, and an inspection of the fire at the end of the test showed in each case a uniform layer with no holes and no banks.

The thickness of the fire was kept as nearly uniform as was practicable, and the grates were shaken as little as possible. The Lump coal proved the most difficult to fire, especially at the high rate of combustion; and in one high rate test with this grade the grates had to be shaken six times. This was unusual, however, and during the entire series the grates were shaken, on the average, only twice during each run. The approximate thicknesses of the fire carried are shown in Table 5.

TABLE 5

APPROXIMATE THICKNESSES OF FIRE CARRIED

SIZE OF COAL	Rate	Approximate Fire Thickness—Inches		
		At the Beginning	Maximum	At the End
Mine Run	Medium	6	9	6
	High	6	12	12
2" x 3" Nut	Medium	5	8	8
	High	7	12	12
3" x 6" Egg	Medium	7	11	10
	High	9	12	10
2" Lump	Medium	7	12	12
	High	9	13	12
2" Screenings	Medium	4	9	7
	High	6	12	10
1¼" Screenings	Medium	4	8	7
	High	5	10	9

VII. TEST CONDITIONS

Owing to the fact that only two sets of conditions as to speed and cut-off were employed throughout the tests, other conditions such as drafts, temperatures, and pressures were also in general quite uniform for all tests at a given rate. The degree of uniformity shown is in a large measure indicative of desirable test conditions and is therefore in some degree significant of the reliance which may b placed upon the determination of those variables, such as evaporativ performance, which constituted the main purpose of the tests.

Fig. 15 and the discussion of the present section are intended t present the more important test conditions and the variation of thes conditions as between different tests and different groups of tests, an as between the medium and high rate tests. Figs. 29 and 30 i Appendix III present graphical logs for tests 2416 and 2405, on a medium and the other a high rate test, during each of which approx imately ten tons of coal were burned. These graphical logs are repre sentative of the degree of uniformity in test conditions which existe throughout individual tests.

Fig. 15 presents in graphical form averages of test conditions fo all grades of coal for both medium and high rate tests. The graph have been so arranged as to show the variation in conditions for dif ferent grades of coal at a given rate, and also to show difference in conditions between medium and high rate tests.

12. *Drafts.*—The two upper graphs of Fig. 15 present average for front-end and firebox draft. The extremes of front-end draft fo the medium rate tests were 2.8 and 3.8 inches of water; the average for all medium rate tests was 3.4 inches of water. The extremes of front-end draft for the high rate tests were 8.4 and 10.1 inches of water and the average for all high rate tests was 9.3 inches of water. The averages for the different grades of coal vary but little from the common average for all of the tests at the corresponding rate. The mean firebox draft for all medium rate tests was 1.6 inches of water and for all high rate tests, 4.2 inches. The averages for the individual tests and for the various grades of coal do not vary greatly from these mean values. The high rate mine run tests show the greatest variation in front-end draft. The high rate Screening tests show a somewhat lower average firebox draft than is shown for the other grades, due probably to the fact that although the rate of combustion was relatively high, the fires were comparatively thin and open.

The data relative to drafts show uniformity of conditions between the tests of a group as well as between the different groups at a given rate of performance. In general the drafts were comparatively low in relation to rate of combustion, indicating satisfactory arrangement of the draft appliances.

13. *Temperatures.*—Front-end temperatures are shown to have been uniform for the medium and for the high rate tests, both by the graphs of Fig. 15 and by the tabulated values. The average firebox temperatures also are shown to have been fairly uniform for the high rate tests and slightly less so for the medium rate tests. The minimum, maximum, and average values of firebox temperature for all medium rate tests were respectively 1735, 2090, and 1893 degrees F.; and the corresponding values for the high rate tests were 2078, 2334, and 2228 degrees F.

14. *Superheat and Branch-pipe Pressure.*—The variations in, and the values for, averages of degrees of superheat and pressure in branch-pipe are shown graphically in Fig. 15. The minimum, maximum, and average values of degrees of superheat for all medium rate tests were respectively 187, 211, and 198 degrees; and the corresponding values for the high rate tests were 207, 265, and 243 degrees. Considerable lack of uniformity is shown by the results, particularly in view of the general uniformity which existed in other test conditions. The branch-pipe pressure for all medium rate tests averaged 179 pounds per square inch, which was 10 pounds lower than the average boiler pressure for the same tests. For the high rate tests the branch-pipe pressure averaged 166 pounds per square inch—22 pounds lower than the corresponding average boiler pressure.

15. *Rate of Combustion and Rate of Evaporation.*—The two lower graphs of Fig. 15 show the average rate of combustion and the average rate of evaporation for the different sizes of coal. Rate of combustion is shown in pounds of coal fired per square foot of grate per hour; and rate of evaporation, in pounds of equivalent evaporation per square foot of heating surface per hour. For the medium rate tests the minimum, maximum, and average values for rate of combustion were, respectively, 45.1, 61.9, and 51.7 pounds of coal per square foot of grate per hour; while for the high rate tests the corresponding values were 99.1, 133.7, and 109.6 pounds. Since the rate of evaporation per square foot of heating surface per hour was maintained approximately constant for all tests at a given rate and since the heating value of all sizes of the coal was about the same, it follows that the various rates of combustion should indicate closely the relative efficiencies with which the coal was burned; and they may be used as rough measures of the relative values of the different grades.

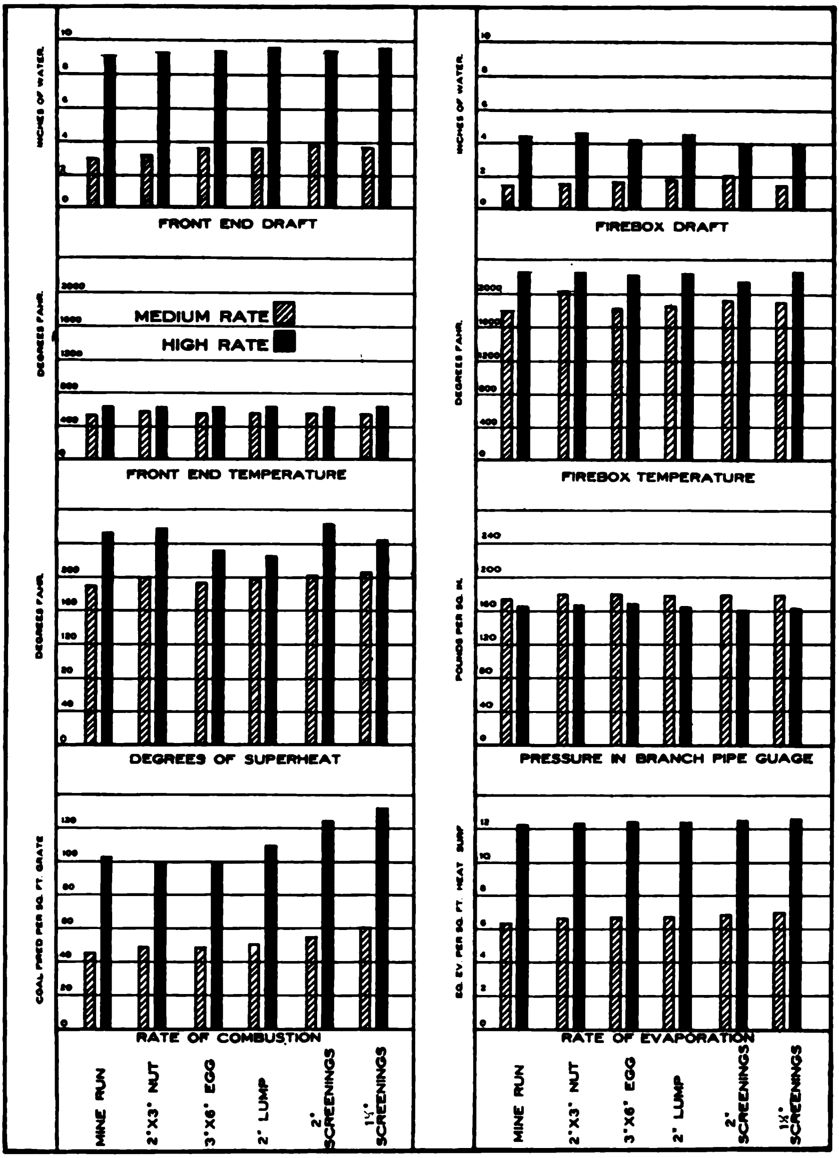


FIG. 15. VARIOUS TEST CONDITIONS, FOR BOTH THE MEDIUM AND HIGH RATE TESTS

VIII. The Results of the Tests

All the data and the results of the tests are set forth in detail in the tables of Appendix III; but for convenience of reference certain data defining the test conditions and some of the results relating to evaporative performance, cinder loss, and heat distribution have been assembled here and are presented in Table 6. In this table the data are divided into twelve groups; two groups for each size of coal tested. For each size the first group pertains to the medium rate tests; the second, to the high rate tests. Averages for each group appear immediately below the data for the individual tests. In Table 6 the column headings and the code item numbers are identical with those in the tables of Appendix III.

16. *Actual Evaporation per Pound of Coal.*—The number of pounds of superheated steam produced for each pound of coal in the condition in which it was fired appears in Column 33 of Table 6; and the number of pounds of steam produced by each pound of dry coal is given in Column 34. The averages of these values for each size of coal are brought together in Table 7, where they appear separately for the medium rate and the high rate tests. Inspection of Table 7 reveals the fact that the relative standing of the various sizes differs when based on coal as fired and when based upon dry coal; and that it differs also between the medium and the high rate tests. Comparisons between the sizes, however, should not be made on the basis of the values of evaporation shown in Table 7; because not only is the moisture content of the coal as fired variable; but during the tests there were slight variations in feed water temperature, in boiler pressure, and in the degree of superheat, which make it impracticable to compare values of actual evaporation in order to determine the relative standing of the coals. Further discussion of Table 7 is omitted for these reasons; the table is presented principally to exhibit the differences between evaporation based on coal as fired and evaporation based on dry coal.

17. *Equivalent Evaporation per Pound of Dry Coal.*—Because of the incidental variations just cited it became necessary to find another basis of comparison. Since the different sizes as they are loaded at the mine contain inherently different amounts of moisture, there would be some justice in trying to base comparisons on coal as loaded at the mine. The significance of this basis is, however, impaired by the fact that the various sizes are seldom or never fired in the same condition

TABLE 6

TEST CONDITIONS AND PRINCIPAL RESULTS

1	2	3	4	5	6	7	8	9	10	11	12
SIZE	Rate	Test Number	Duration of Test in Hours	Speed in Miles per Hour	Cut-off, Per Cent of Stroke	Draw-bar Pull, lb.	Pressure lb. per sq. in.		Temperatures Degrees F.		
							Boiler Gauge	Branch Pipe Gauge	Front-end	Branch Pipe	Fire-Box
		Code Item ☞	**345**	**353**	**499**	**487**	**380**	**383**	**367**	**370**	**374**
Mine Run	Medium	2400	3.62	18.9	....	21970	190.4	179	535	573	1735
		2401	6.28	18.9	34.0	21727	190.0	172	535	566	1835
		2402	5.20	19.0	33.0	21822	190.2	174	539	564	1812
		Average	**5.03**	**18.9**	**33.5**	**21840**	**190.2**	**175**	**536**	**568**	**1794**
	High	2405	2.72	25.5	54.3	28771	187.8	168	627	628	2271
		2406	2.68	25.6	53.7	28718	187.8	167	631	631	2334
		2429	1.92	25.7	55.9	28672	189.4	164	624	618	2140
		Average	**2.44**	**25.6**	**54.6**	**28720**	**188.3**	**166**	**627**	**626**	**2248**
2″x 3″ Nut	Medium	2408	3.77	19.0	33.0	22490	189.0	178	595	589	2090
		2409	2.33	18.9	32.4	22411	188.5	177	588	582	2034
		2410	4.33	19.0	31.5	22417	186.2	181	570	569	2008
		2426	4.50	18.9	31.7	22640	189.9	182	555	572	1967
		Average	**3.73**	**19.0**	**32.2**	**22490**	**188.4**	**180**	**577**	**578**	**1025**
	High	2412	2.67	25.8	....	28958	187.1	168	607	629	2293
		2413	3.00	25.7	55.7	29100	187.1	168	611	632	2267
		2414	2.00	25.7	59.3	29128	187.5	168	631	634	2174
		Average	**2.56**	**25.7**	**57.5**	**29062**	**187.2**	**168**	**616**	**632**	**2248**
3″x 6″ Egg	Medium	2415	3.50	18.9	32.4	22840	189.9	180	543	576	1808
		2416	5.83	18.9	33.3	23115	189.5	180	540	574	1801
		2423	4.00	18.8	32.2	22533	190.0	182	539	571	....
		Average	**4.44**	**18.9**	**32.6**	**22829**	**189.8**	**181**	**541**	**574**	**1805**
	High	2420	2.00	25.7	57.2	29046	190.1	171	588	610	2210
		2422	2.17	25.9	58.2	29030	189.7	170	634	590	2278
		2424	1.98	25.8	56.6	29104	190.1	170	626	617	2183
		Average	**2.05**	**25.8**	**57.3**	**29060**	**190.0**	**170**	**616**	**606**	**2224**
2″ Lump	Medium	2417	4.00	18.9	36.3	23026	189.9	180	546	578	1838
		2418	5.83	19.0	33.5	23085	190.1	180	545	578	1857
		2419	3.67	19.0	32.7	22983	190.0	180	553	578	1849
		Average	**4.50**	**19.0**	**34.2**	**23031**	**190.0**	**180**	**548**	**578**	**1848**
	High	2425	1.00	25.7	56.0	28530	190.0	168	618	595	2178
		2427	1.50	25.8	55.7	27909	183.4	162	625	578	2308
		2428	1.83	25.9	55.1	28441	186.8	166	635	603	2277
		2442	2.00	25.5	56.5	29266	188.9	166	637	616	2192
		Average	**1.56**	**25.7**	**55.8**	**28537**	**187.3**	**166**	**629**	**598**	**2239**
2″ Screenings	Medium	2430	2.62	19.0	32.6	22906	188.6	181	549	583	2010
		2434	3.13	18.7	33.6	23091	190.0	181	541	584	1817
		2435	0.97	19.0	34.9	23268	191.1	182	549	578	1936
		Average	**2.24**	**18.9**	**33.7**	**23088**	**189.9**	**181**	**548**	**582**	**1921**
	High	2436	1.35	25.5	56.8	27976	185.3	161	631	634	2078
		2437	1.50	25.7	56.9	28938	189.1	162	634	637	2194
		Average	**1.43**	**25.6**	**56.9**	**28457**	**187.2**	**162**	**633**	**636**	**2136**
1¼″ Screenings	Medium	2431	1.77	19.0	33.9	22332	184.5	178	551	589	2003
		2432	1.87	19.0	34.0	22912	189.1	181	544	591	1798
		2433	3.10	18.9	33.2	22588	187.7	180	543	572	1874
		Average	**2.25**	**19.0**	**33.7**	**22611**	**187.1**	**180**	**548**	**584**	**1892**
	High	2440	1.50	25.5	58.2	29061	186.6	163	634	604	2273
		2441	1.50	25.6	55.6	29392	191.0	166	639	629	2234
		Average	**1.50**	**25.6**	**56.9**	**29227**	**188.8**	**165**	**637**	**617**	**2254**

TABLE 6 (Continued)

TEST CONDITIONS AND PRINCIPAL RESULTS

1	2	3	13	14	15	16	17	18	19	20	21
			Draft, in. of Water				Coal as Fired, lb.			Dry Coal, lb.	
SIZE	Rate	Test Number	Front-end Front of Diaphragm	Fire-box	Ash-pan	Degrees of Superheat	Total	Per Hour	Per Hour per Sq. Ft. of Grate Surface	Per Hour	Per Hour per Sq. Ft. of Grate Surface
		Code Item ☞	394	396	397	409	418	......	....	626	627
Mine Run	Medium	2400	2.8	1.2	0.2	194	11399	3151	45.1	2895	41.5
		2401	2.8	1.5	0.2	190	20000	3183	45.6	2934	42.0
		2402	3.0	1.6	0.2	187	17000	3269	46.8	3005	43.1
		Average	**2.9**	**1.4**	**0.2**	**190**	**16133**	**3201**	**45.8**	**2945**	**42.2**
	High	2405	8.4	4.2	0.4	254	20000	7361	105.5	6753	96.8
		2405	8.6	4.3	0.4	257	18630	6944	99.5	6377	91.4
		2429	10.1	4.5	0.4	246	14000	7303	104.6	6690	95.9
		Average	**9.0**	**4.3**	**0.4**	**252**	**17543**	**7203**	**103.2**	**6607**	**94.7**
2"x3" Nut	Medium	2408	3.0	1.4	0.2	210	12955	3439	49.3	3146	45.1
		2409	2.9	1.3	0.2	204	7808	3347	48.0	3054	43.8
		2410	2.9	1.4	0.2	189	14731	3400	48.7	3102	44.4
		2426	3.6	1.8	0.1	192	16310	3624	51.9	3348	48.0
		Average	**3.1**	**1.5**	**0.2**	**199**	**12951**	**3453**	**49.5**	**3163**	**45.3**
	High	2412	9.2	4.6	0.5	255	18683	7005	100.4	6382	91.4
		2413	9.2	4.4	0.5	258	20811	6937	99.4	6313	90.4
		2414	9.3	4.5	0.5	260	13884	6942	99.5	6324	90.6
		Average	**9.2**	**4.5**	**0.5**	**258**	**17793**	**6961**	**99.8**	**6340**	**90.8**
3"x6" Egg	Medium	2415	3.6	1.8	0.2	197	11888	3397	48.7	3107	44.5
		2416	3.6	1.7	0.2	195	19915	3414	48.9	3101	44.4
		2423	3.3	1.4	0.2	191	13520	3380	48.4	3079	44.1
		Average	**3.5**	**1.6**	**0.2**	**194**	**15108**	**3397**	**48.7**	**3096**	**44.3**
	High	2420	9.5	4.3	0.5	235	13882	6941	99.4	6333	90.7
		2422	9.2	4.0	0.5	215	14996	6920	99.1	6315	90.5
		2424	9.3	4.1	0.5	242	14000	7060	101.2	6438	92.2
		Average	**9.3**	**4.1**	**0.5**	**231**	**14293**	**6974**	**99.9**	**6362**	**91.1**
2" Lump	Medium	2417	3.6	1.7	0.2	199	13753	3438	49.3	3118	44.7
		2418	3.5	1.7	0.2	199	20537	3521	50.5	3173	45.5
		2419	3.5	1.7	0.2	199	13344	3639	52.1	3289	47.1
		Average	**3.5**	**1.7**	**0.2**	**199**	**15878**	**3533**	**50.6**	**3193**	**45.8**
	High	2425	9.3	4.3	0.5	221	7499	7499	107.4	6850	98.1
		2427	9.3	4.3	0.4	207	11775	7850	112.5	7133	102.2
		2428	9.4	4.4	0.3	230	14122	7704	110.4	6992	100.2
		2442	10.0	4.6	0.5	243	15279	7640	109.5	6951	99.6
		Average	**9.5**	**4.4**	**0.4**	**225**	**12169**	**7673**	**110.0**	**6982**	**100.0**
2" Screenings	Medium	2430	3.8	1.9	0.2	203	10000	3821	54.7	3460	49.6
		2434	3.6	......	0.2	204	11950	3814	54.6	3455	49.5
		2435	3.7	2.1	0.2	198	3822	3952	56.6	3597	51.5
		Average	**3.7**	**2.0**	**0.2**	**202**	**8591**	**3862**	**55.3**	**3504**	**50.2**
	High	2436	9.4	3.7	0.4	263	11556	8560	122.6	7771	111.3
		2437	9.2	3.8	0.3	265	13254	8836	126.6	8027	115.0
		Average	**9.3**	**3.8**	**0.4**	**264**	**12405**	**8698**	**124.6**	**7899**	**113.2**
1½" Screenings	Medium	2431	3.5	1.9	0.2	210	7635	4321	61.9	3960	56.7
		2432	3.6	1.1	0.2	211	7813	4185	60.0	3818	54.7
		2433	3.6	1.3	0.2	193	13218	4264	61.1	3899	55.9
		Average	**3.6**	**1.4**	**0.2**	**205**	**9555**	**4257**	**61.0**	**3892**	**55.8**
	High	2440	9.5	4.0	0.5	232	14000	9333	133.7	8487	121.6
		2441	9.4	3.7	0.5	256	13750	9167	131.3	8183	117.2
		Average	**9.5**	**3.9**	**0.5**	**244**	**13875**	**9250**	**132.5**	**8335**	**119.4**

TABLE 6 (Continued)

TEST CONDITIONS AND PRINCIPAL RESULTS

1	2	3	22	23	24	25	26	27	28	29	30
SIZE	Rate	Test Number	B.t.u. per lb. of		Cinder Loss				Indicated Horse-Power	Drawbar Horse-Power	Superheated Steam per I.H.P. Hour, lb.
			Dry-Coal	Stack Cinders	Per Hour, lb.	Per Cent of Coal as Fired	Per Cent of Dry Coal	Per Cent of B.t.u. in Coal Fired			
		Code Item	458	462	......	......	427	888	711	743	740
Mine Run	Medium	2400	12983	8399	99	3.2	3.4	2.2	....	1108.6	......
		2401	13012	8563	94	3.0	3.2	2.1	1224.5	1095.2	18.18
		2402	12929	8570	102	3.1	3.4	2.2	1223.6	1104.6	18.74
		Average	**12975**	**8511**	**98**	**3.1**	**3.3**	**2.2**	**1224.1**	**1102.8**	**18.46**
	High	2405	12811	11081	709	9.6	10.5	9.1	2157.7	1954.4	19.63
		2406	12933	11030	588	8.5	9.2	7.9	2151.5	1963.8	19.40
		2429	12888	10921	651	8.9	9.7	8.2	2191.0	1965.2	19.65
		Average	**12877**	**11011**	**649**	**9.0**	**9.8**	**8.4**	**2166.7**	**1961.1**	**19.56**
2″x 3″ Nut	Medium	2408	13118	8023	63	1.8	2.0	1.2	1293.6	1138.4	17.93
		2409	13102	7585	72	2.2	2.4	1.4	1286.5	1129.7	17.98
		2410	13023	8231	72	2.1	2.3	1.5	1280.9	1133.5	17.93
		2426	12983	8458	107	3.0	3.2	2.1	1313.7	1139.9	18.72
		Average	**13057**	**8074**	**79**	**2.3**	**2.5**	**1.6**	**1293.7**	**1135.4**	**18.14**
	High	2412	13081	10728	413	5.9	6.5	5.3	....	1988.7	......
		2413	13176	10822	397	5.7	6.3	5.2	2200.3	1993.0	19.34
		2414	13090	10634	390	5.6	6.2	5.0	2221.8	1994.9	19.00
		Average	**13116**	**10728**	**400**	**5.7**	**6.3**	**5.2**	**2211.1**	**1992.2**	**19.17**
3″x 6″ Egg	Medium	2415	13273	7987	78	2.3	2.5	1.5	1313.7	1150.1	18.01
		2416	13122	7999	76	2.2	2.5	1.5	1324.3	1167.6	17.86
		2423	13282	8329	70	2.1	2.3	1.4	1291.1	1131.0	18.41
		Average	**13226**	**8105**	**75**	**2.2**	**2.4**	**1.5**	**1309.7**	**1149.6**	**18.09**
	High	2420	13198	10771	500	7.2	7.9	6.5	2188.7	1991.6	19.83
		2422	13345	11234	468	6.8	7.4	6.2	2214.3	2001.4	19.40
		2424	13214	10584	538	7.6	8.4	6.7	2220.0	2003.4	19.63
		Average	**13252**	**10863**	**502**	**7.2**	**7.9**	**6.5**	**2207.7**	**1998.8**	**19.62**
2″ Lump	Medium	2417	13043	7713	71	2.1	2.3	1.4	1323.2	1160.7	17.92
		2418	13086	7574	68	1.9	2.1	1.2	1329.6	1168.6	17.73
		2419	12836	7106	85	2.3	2.6	1.4	1321.0	1162.2	18.31
		Average	**12988**	**7464**	**75**	**2.1**	**2.3**	**1.3**	**1324.6**	**1163.8**	**17.99**
	High	2425	13205	10917	545	7.3	8.0	6.6	2201.4	1957.8	19.61
		2427	12958	10849	602	7.7	8.4	7.1	2169.7	1919.6	19.68
		2428	13061	10829	523	6.8	7.5	6.2	2200.5	1963.8	19.95
		2442	12974	10415	595	7.8	8.6	6.9	2198.5	1989.6	20.24
		Average	**13050**	**10753**	**566**	**7.4**	**8.4**	**6.7**	**2192.5**	**1957.7**	**19.87**
2″ Screenings	Medium	2430	12748	9407	315	8.3	9.1	6.7	1304.9	1160.7	18.02
		2434	12782	9569	338	8.9	9.8	7.3	1314.5	1152.2	18.00
		2435	12710	9113	372	9.4	10.4	7.4	1365.2	1179.1	18.25
		Average	**12747**	**9363**	**342**	**8.9**	**9.8**	**7.1**	**1328.2**	**1164.0**	**18.09**
	High	2436	12769	10611	1127	13.2	14.5	12.1	2126.3	1905.6	19.86
		2437	12625	11018	1316	14.9	16.4	14.3	2243.5	1980.4	19.33
		Average	**12697**	**10815**	**1222**	**14.1**	**15.5**	**13.2**	**2184.9**	**1943.0**	**19.60**
1½″ Screenings	Medium	2431	12929	10505	579	13.4	14.6	11.9	1329.9	1133.4	18.32
		2432	12650	10157	551	13.2	14.4	11.6	1343.1	1161.6	18.22
		2433	12793	10784	461	10.8	11.8	10.0	1310.2	1141.0	18.71
		Average	**12791**	**10482**	**530**	**12.5**	**13.6**	**11.2**	**1327.7**	**1145.3**	**18.42**
	High	2440	12692	10870	1513	16.2	17.8	15.3	2215.6	1977.2	19.29
		2441	12492	11203	1457	15.9	17.8	16.0	2231.6	2006.7	19.48
		Average	**12592**	**11037**	**1485**	**16.1**	**17.8**	**15.7**	**2223.6**	**1992.0**	**19.39**

TABLE 6 (Concluded)

TEST CONDITIONS AND PRINCIPAL RESULTS

1	2	3	31	32	33	34	35	36	37	38	39
			Superheated Steam, lb.				Equivalent Evaporation, lb.				
Size	Rate	Test Number	Per Hour	Per Sq.Ft. of Heating Surface per Hour	Per Pound of Coal as Fired	Per Pound of Dry Coal	Per Hour	Per Sq. Ft. Heating Surface per Hour	Per Pound of Dry Coal	B.t.u. Absorbed by Boiler per Pound of Dry Coal	Boiler Efficiency Per Cent
		Code Item ☞	......	......	......	......	645	648	658	......	666
Mine Run	Medium	2400	22566	4.84	7.16	7.79	29764	6.39	10.28	9989	76.89
		2401	22328	4.79	7.01	7.61	29451	6.32	10.04	9756	74.95
		2402	22970	4.93	7.02	7.64	30183	6.48	10.04	9756	75.46
		Average	**22621**	**4.85**	**7.06**	**7.63**	**29799**	**6.40**	**10.12**	**9834**	**75.77**
	High	2405	42176	9.05	5.73	6.24	57022	12.24	8.44	8201	64.08
		2406	41946	9.00	6.04	6.58	56795	12.19	8.91	8658	66.93
		2429	42854	9.20	5.87	6.41	57767	12.40	8.64	8395	65.10
		Average	**42325**	**9.08**	**5.88**	**6.41**	**57195**	**12.28**	**8.66**	**8418**	**65.37**
2″x 3″ Nut	Medium	2408	23327	5.00	6.78	7.42	31048	6.66	9.87	9591	73.11
		2409	23254	4.99	6.95	7.61	30881	6.63	10.11	9824	75.02
		2410	23059	4.95	6.78	7.43	30461	6.54	9.82	9542	73.05
		2426	24808	5.33	6.85	7.41	32796	7.04	9.80	9523	73.33
		Average	**23612**	**5.07**	**6.34**	**7.47**	**31297**	**6.72**	**9.90**	**9620**	**73.63**
	High	2412	42964	9.22	6.13	6.73	58130	12.47	9.11	8852	67.67
		2413	42720	9.17	6.16	6.77	57929	12.43	9.18	8920	67.67
		2414	42209	9.06	6.08	6.67	57236	12.28	9.05	8794	67.15
		Average	**42631**	**9.15**	**6.12**	**6.72**	**57765**	**12.39**	**9.11**	**8855**	**67.50**
3″x 6″ Egg	Medium	2415	23944	5.14	7.05	7.71	31678	6.80	10.20	9911	74.66
		2416	23788	5.11	6.97	7.67	31448	6.75	10.14	9853	75.09
		2423	23970	5.14	7.09	7.78	31665	6.79	10.28	9989	75.25
		Average	**23901**	**5.13**	**7.04**	**7.72**	**31597**	**6.78**	**10.21**	**9918**	**75.00**
	High	2420	43219	9.28	6.22	6.82	58043	12.46	9.16	8901	67.46
		2422	42881	9.20	6.20	6.79	57160	12.27	9.05	8794	65.92
		2424	43302	9.29	6.13	6.73	58328	12.52	9.06	8804	66.61
		Average	**43134**	**9.26**	**6.18**	**6.78**	**57844**	**12.42**	**9.09**	**8833**	**66.66**
2″ Lump	Medium	2417	23906	5.13	6.96	7.67	31652	6.79	10.15	9863	75.68
		2418	23778	5.10	6.75	7.49	31458	6.75	9.91	9630	73.57
		2419	24340	5.23	6.69	7.40	32227	6.92	9.80	9523	74.21
		Average	**24008**	**5.15**	**6.80**	**7.52**	**31779**	**6.82**	**9.95**	**9672**	**74.49**
	High	2425	42889	9.21	5.72	6.26	57214	12.28	8.35	8114	61.47
		2427	42254	9.07	5.38	5.92	56071	12.03	7.86	7638	58.92
		2428	44136	9.47	5.73	6.31	58403	12.53	8.35	8114	62.14
		2442	44910	9.64	5.88	6.46	60539	12.99	8.71	8464	65.19
		Average	**43547**	**9.35**	**5.68**	**6.24**	**58057**	**12.46**	**8.32**	**8083**	**61.93**
2″ Screenings	Medium	2430	24122	5.18	6.31	6.97	32010	6.87	9.25	8988	70.55
		2434	24014	5.15	6.30	6.95	31914	6.85	9.24	8979	70.24
		2435	25147	5.40	6.36	6.99	33270	7.14	9.25	8988	70.75
		Average	**24427**	**5.24**	**6.32**	**6.97**	**32398**	**6.95**	**9.25**	**8985**	**70.51**
	High	2436	42287	9.07	4.94	5.45	57468	12.33	7.40	7191	56.25
		2437	43981	9.44	4.98	5.48	59814	12.84	7.45	7239	57.35
		Average	**43134**	**9.26**	**4.96**	**5.47**	**58641**	**12.59**	**7.43**	**7215**	**56.80**
1¼″ Screenings	Medium	2431	24907	5.34	5.76	6.29	33102	7.10	8.36	8123	62.81
		2432	24714	5.30	5.90	6.47	32870	7.05	8.61	8366	66.08
		2433	24933	5.35	5.85	6.39	32961	7.07	8.45	8211	64.21
		Average	**24851**	**5.33**	**5.84**	**6.38**	**32978**	**7.07**	**8.47**	**8233**	**64.37**
	High	2440	43186	9.27	4.63	5.09	57956	12.44	6.83	6637	52.28
		2441	43941	9.43	4.80	5.37	59540	12.78	7.28	7074	56.64
		Average	**43564**	**9.35**	**4.72**	**5.23**	**58748**	**12.61**	**7.06**	**6856**	**54.46**

(as regards moisture) in which they were loaded, nor were they in the same condition in this instance; and furthermore the necessary mine samples were available for only three of the six sizes tested. Final comparisons were consequently drawn only on the usual basis of dry coal. The use of the customary "equivalent evaporation from and at 212 degrees" eliminates the effect of the remaining variations in test conditions; and the final comparison of the grades was therefore made by the use of the values of this equivalent evaporation per pound of dry coal.

TABLE 7

THE ACTUAL EVAPORATION PER POUND OF COAL AS FIRED AND ALSO PER POUND OF DRY COAL

SIZE OF COAL	Actual Evaporation per lb. of Coal as Fired—lb.		Actual Evaporation per lb. of Dry Coal—lb.	
	At the Medium Rate of Evaporation	At the High Rate of Evaporation	At the Medium Rate of Evaporation	At the High Rate of Evaporation
1	2	3	4	5
Mine Run	7.06	5.88	7.68	6.41
2″ x 3″ Nut	6.84	6.12	7.47	6.72
3″ x 6″ Egg	7.04	6.18	7.72	6.78
2″ Lump	6.80	5.68	7.52	6.24
2″ Screenings	6.32	4.96	6.97	5.47
1¼″ Screenings	5.84	4.72	6.38	5.23

These values of equivalent evaporation per pound of dry coal are given for each test in Column 37 of Table 6. Inspection of these figures discloses great uniformity among the values applying to each size and each rate. Only in the case of the high rate tests with the 2-inch lump coal is there any considerable variation between the equivalent evaporation values for the individual tests; and even in this group the maximum variation from the average is only 5½ per cent. In view of this uniformity we are entirely warranted in using the average values for the various groups and in basing conclusions upon them. These averages of equivalent evaporation per pound of dry coal are therefore assembled in Table 8 together with the averages of the rate of evaporation per square foot of heating surface per hour taken from Column 36 of Table 6. Table 8 embodies consequently the final direct results of the tests.

In Table 8 the coals are arranged in the order of the evaporation at the medium rate as given there in Column 2. The egg coal heads the list with an equivalent evaporation of 10.21 pounds per pound of dry

TABLE 8

THE EQUIVALENT EVAPORATION PER POUND OF DRY COAL FOR BOTH THE MEDIUM AND THE HIGH RATE TESTS

SIZE OF COAL	For the Medium Rate Tests		For the High Rate Tests	
	Equivalent Evaporation per lb. of Dry Coal lb.	Equivalent Evaporation per Hour per sq. ft. of Heating Surface lb.	Equivalent Evaporation per lb. of Dry Coal lb.	Equivalent Evaporation per Hour per sq. ft. of Heating Surface lb.
1	2	3	4	5
3" x 6" Egg	10.21	6.78	9.09	12.42
Mine Run	10.12	6.40	8.66	12.28
2" Lump	9.95	6.82	8.32	12.46
2" x 3" Nut	9.90	6.72	9.11	12.39
2" Screenings	9.25	6.95	7.43	12.59
1¼" Screenings	8.47	7.07	7.06	12.61

coal followed by the other grades in the order in which they appear in the table. For the high rate tests the nut coal gave the best performance, namely an equivalent evaporation of 9.11 pounds per pound of dry coal, while the other sizes stand in the order in which they are cited in the table. These relations stand out more clearly in Fig 16 which has been prepared by plotting values of equivalent evaporation and rate of evaporation given in Table 8. In Fig. 16 the vertical distances represent equivalent evaporation per pound of dry coal, whereas the horizontal distances represent the pounds of equivalent evaporation per hour on each square foot of heating surface. For the 3-inch by 6-inch egg coal these quantities are, for the medium rate tests 10.21 pounds and 6.78 pounds, respectively; and for the high rate tests 9.09 pounds and 12.42 pounds, respectively (see Table 8). These pairs of values are plotted in Fig. 16 where they appear as the two points which define the line marked 3-inch by 6-inch Egg. These points are connected by a straight line, which implies the assumption that the equivalent evaporation varies regularly and directly with the rate of evaporation. While there are, in this series, no tests at intermediate rates to support this assumption, it is amply warranted by the results of numerous other locomotive boiler tests. The other lines in Fig. 16 are similarly plotted from values given in Table 8, and they define the performance of the other five sizes.

Inspection of Fig. 16 reveals, as usual, for all grades a sharp decrease in evaporation as the rate of evaporation increases. The rate

of this decrease is nearly alike for all sizes except the
3-inch nut for which it is roughly one-half of that for the ot
This change in evaporation with rate of evaporation makes it
to reduce the values of evaporation to a common rate before
final comparisons between the various grades. To effect this
the rates of evaporation for all the medium rate tests show
umn 3 of Table 8 have been averaged and this average—6.7
per square foot of heating surface per hour—has been re

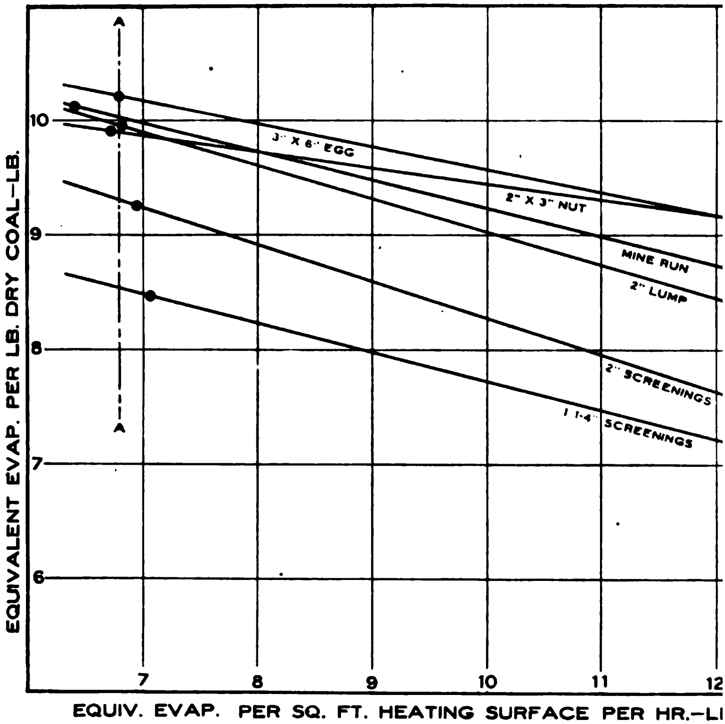


Fig. 16. The Relation Between Equivalent Evaporation Per Pou
Coal and the Rate of Evaporation, for Each Size of Coal T

by the vertical line *AA* in Fig. 16. Similarly the average hi
12.46 pounds per square foot of heating surface per hour–
from Column 5 of Table 8 and is defined by the line *B*.
figure. If now in Fig. 16 we measure off the vertical distanc
at the points where this line is intersected by the perform
for the various sizes, we obtain six values of equivalent ev

per pound of dry coal, one for each size, and all pertaining to the common medium rate of evaporation defined by the line *AA*, namely, 6.79 pounds per square foot of heating surface per hour. These values are shown in Column 2 of Table 9 and since they pertain to the same rate of evaporation they are rigidly comparable. In like manner the evaporation values defined by the intersections with the line *BB* are given in Column 4 of Table 9, and pertain to the common high rate—12.46 pounds per square foot of heating surface per hour.

TABLE 9

THE RELATIVE STANDING OF THE VARIOUS SIZES, BASED ON CORRECTED VALUES OF THE EQUIVALENT EVAPORATION PER POUND OF DRY COAL

(This table gives the values of equivalent evaporation per pound of dry coal, corrected for a common medium rate of evaporation of 6.79 pounds per square foot of heating surface per hour, and for a common high rate of evaporation of 12.46 pounds per square foot of heating surface per hour)

GRADE OF COAL	For the Common **Medium** Rate of Evaporation—6.79 lb. per sq. ft. of Heating Surface per Hour		For the Common **High** Rate of Evaporation—12.46 lb. per sq. ft. of Heating Surface per Hour	
	Equivalent Evaporation per lb. of Dry Coal lb.	Relative Values, Based on Mine Run	Equivalent Evaporation per lb. of Dry Coal lb.	Relative Values, Based on Mine Run
1	2	3	4	5
3" x 6" Egg	10.21	1.02	9.08	1.05
Mine Run	10.02	1.00	8.62	1.00
2" Lump	9.96	0.99	8.32	0.97
2" x 3" Nut	9.89	0.98	9.10	1.06
2" Screenings	9.30	0.93	7.47	0.87
1¼" Screenings	8.54	0.85	7.10	0.82

Table 9 presents therefore average values of equivalent evaporation per pound of dry coal for each of the sizes of coal—first for a common medium rate of evaporation in Column 2, and next for a common high rate of evaporation in Column 4. These are the final results of the tests, and they may be compared to determine the relative value of the various sizes. Such a comparison of the values in Column 2 shows that when the boiler was worked at the medium rate the 3-inch by 6-inch egg coal gave the highest evaporation, with the other sizes following in the order in which they appear in the table; whereas at the high rate the 2-inch by 3-inch nut coal gave the highest evaporation followed by the egg, mine run, lump, 2-inch screenings, and 1¼-inch screenings in the order named. Further comparison is more conveniently made upon a percentage basis, for which purpose the performance of the mine run is taken as unity or 100 per cent, and the other sizes are represented in Columns 3 and 5 of Table 9 by numbers which define the relation of their performance to that of the mine run.

At the *medium rate* the four larger grades gave nearly the same performance, the maximum difference among them being but 4 per cent. The steam production per pound of egg coal was 2 per cent greater than with the mine run, while with the lump and the nut it was respectively 1 per cent and 2 per cent less than with mine run. The performance with 2-inch screenings was 7 per cent less and with 1¼-inch screenings 15 per cent less than with mine run. If we assume that mine run coal on the tender was worth $2.00 per ton, the relative worth on the tender of the other sizes during the medium rate tests was:

3-inch x 6-inch Egg . .	$2.04	2-inch Screenings . . .	$1.86
2-inch Lump	1.98	1¼-inch Screenings . .	1.70
2-inch x 3-inch Nut . .	1.96		

At the *high rate* the 2-inch by 3-inch nut coal gave the best performance, producing 6 per cent more steam than the mine run; the 3-inch by 6-inch egg came next with an evaporation 5 per cent more than that of the mine run; while the 2-inch lump evaporated 3 per cent less. At this rate of evaporation the 2-inch screenings and the 1¼-inch screenings produced per pound respectively 13 per cent and 18 per cent less steam than the mine run. If we again assume that mine run was worth on the tender $2.00 per ton, the relative worth of the other sizes during the high rate tests was as follows:

2-inch x 3-inch Nut . .	$2.12	2-inch Screenings . . .	$1.74
3-inch x 6-inch Egg . .	2.10	1¼-inch Screenings . .	1.64
2-inch Lump	1.94		

The facts presented in Table 9 and in the foregoing discussion are graphically presented in Figs. 17 and 18. Fig. 17 shows the relative steam producing capacity or the relative values of the six different sizes of fuel when used at the medium rate of evaporation; and Fig. 18 presents these relations for the high rate of evaporation. These two figures and Table 9 embody the final and principal results of the whole test series.

While the differences in performance of the sizes is due in some measure to inherent variations in heating value and in ash content, these variations are too small to account fully for the difference in

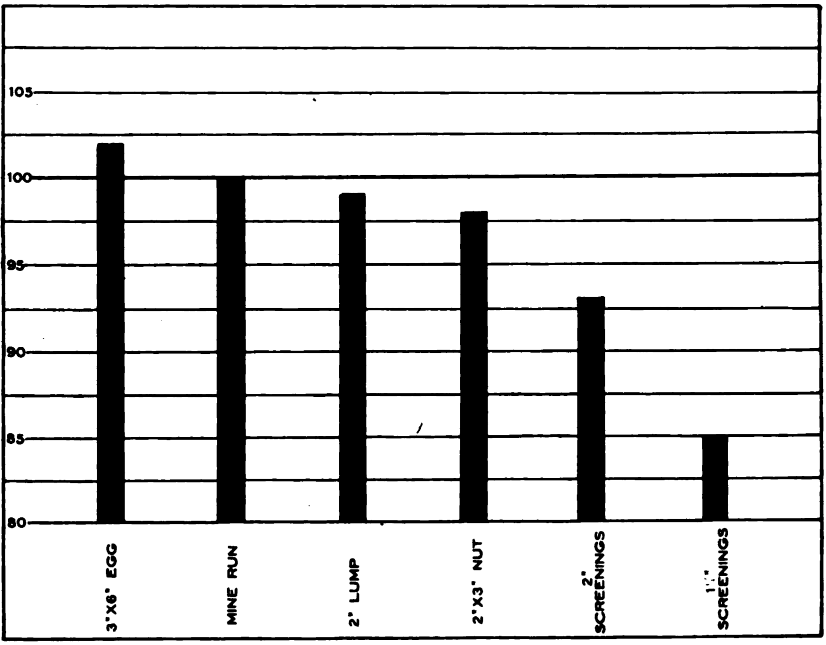


FIG. 17. THE RELATIVE EVAPORATIVE EFFICIENCIES OF THE COALS, FOR THE MEDIUM RATE TESTS

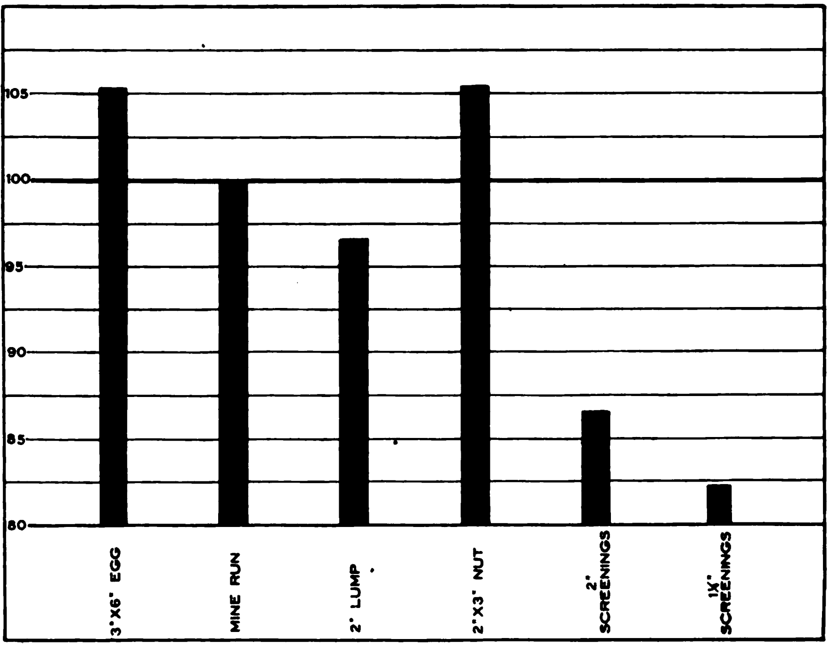


FIG. 18. THE RELATIVE EVAPORATIVE EFFICIENCIES OF THE COALS, FOR THE HIGH RATE TESTS

performance, nor is an explanation on these grounds applicable to all of the sizes. The difference in performance appears to be due chiefly to the variations in cinder loss and in the conditions of combustion which it was possible to maintain with the different sizes. This conclusion is supported by the discussion of cinder losses and of the heat distribution which follows in the next two sections.

18. *Cinder Losses.*—Information relative to the losses due to cinders passing out through the stack is given in Fig. 19. In considering the cinder losses as here presented it should be borne in mind that all of the coal tested was of one kind, that is, it came from one mine. Coals possessing other physical characteristics might show somewhat different results as to cinder losses under the conditions of the tests here considered. It should also be remembered that for a given rate, medium or high, the draft was, for all grades of coal, practically constant as shown in Fig. 15.

Fig. 19 shows the amount of the stack losses when the weight of the cinders collected from the stack is expressed as a percentage of the weight of the dry coal fired, and also the amount of such loss when the heat content of the cinders collected from the stack is expressed as a percentage of the British thermal units in the coal fired. The loss when expressed as per cent of B. t. u. is numerically less than when expressed as per cent of weight of dry coal, due to the fact that the cinders do not have so high a heat value per pound as the coal from which they originate. Also, due to the fact that cinders produced at high rates of combustion have higher heating values than cinders produced at low rates of combustion, the differences between percentages for medium rate and high rate tests are greater when expressed in terms of heat units than when expressed in terms of dry coal. The average heating value of the stack cinders for all medium rate tests was 8635 B. t. u. and the average value for all high rate tests was 10854 B. t. u. Column 23, Table 6, shows the heating value of stack cinders for each test and the average values for each of the twelve groups of tests. The heating values of the cinders from the medium rate tests with screenings were higher than corresponding values from other grades of coal.

In Fig. 19 it will be seen that, during the medium rate tests, from 2.3 to 13.6 pounds of cinders were collected from the stack for each 100 pounds of dry coal fired; while for the high rate tests from 6.3 to 17.8 pounds were collected for each 100 pounds of coal. The

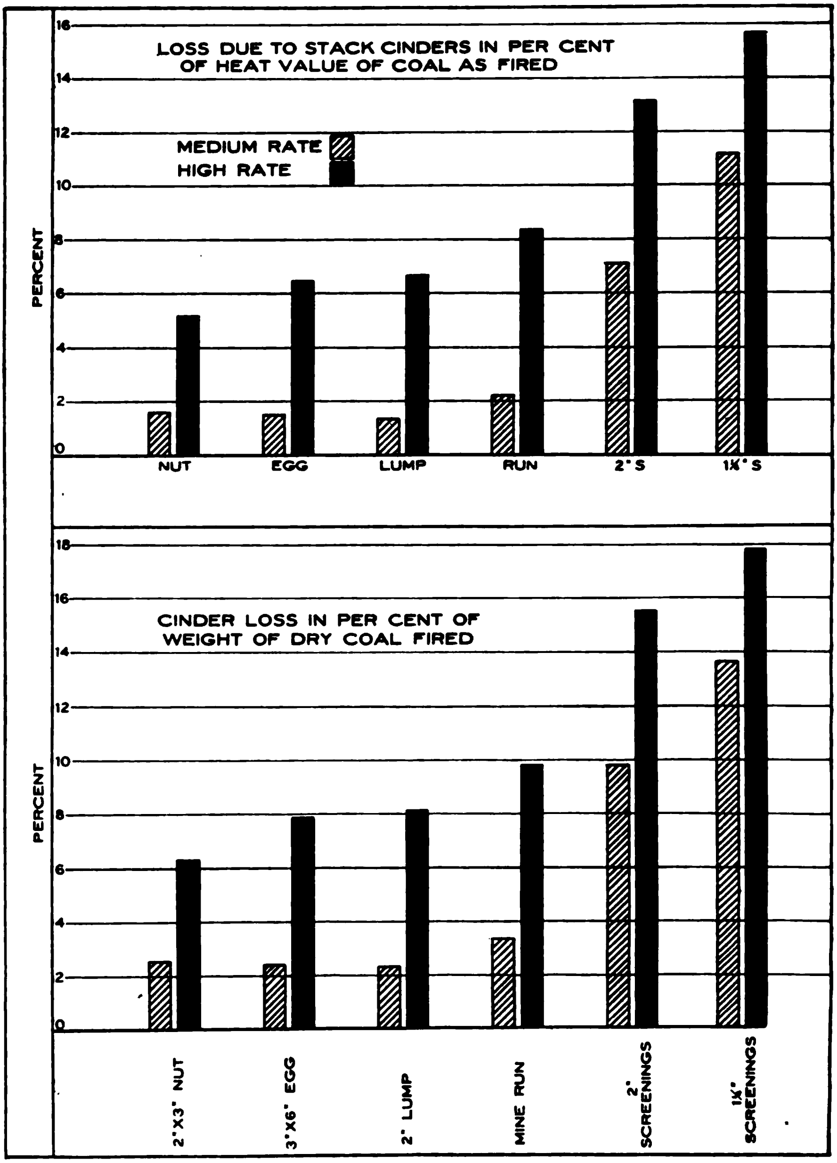


FIG. 19. THE CINDER LOSSES, EXPRESSED IN PER CENT OF THE HEAT IN THE COAL AND AS PER CENT OF THE WEIGHT OF THE DRY COAL

screened coals in all cases produced fewer cinders than the mine run coal; and the screenings produced a materially greater quantity of cinders than any of the larger sizes.

When the losses are expressed as B. t. u. percentages, Fig. 19 shows that for the medium rate tests the loss on account of stack cinders was smallest in the case of the lump coal, amounting to 1.3 per cent of the heat content of the coal fired. The corresponding losses for the egg, nut and mine run coals were 1.5, 1.6, and 2.2 per cent, respectively. The loss, during medium rate tests, for the 2-inch screenings was 7.1 per cent and for the 1¼-inch screenings, 11.2 per cent. The average loss from the screenings was roughly five times as great as the average loss from the larger coals during the medium rate tests. For the high rate tests the smallest heat loss due to stack cinders occurred with the nut coal. The average losses for nut, egg, lump and mine run are 5.2, 6.5, 6.7, and 8.4 per cent, respectively. The

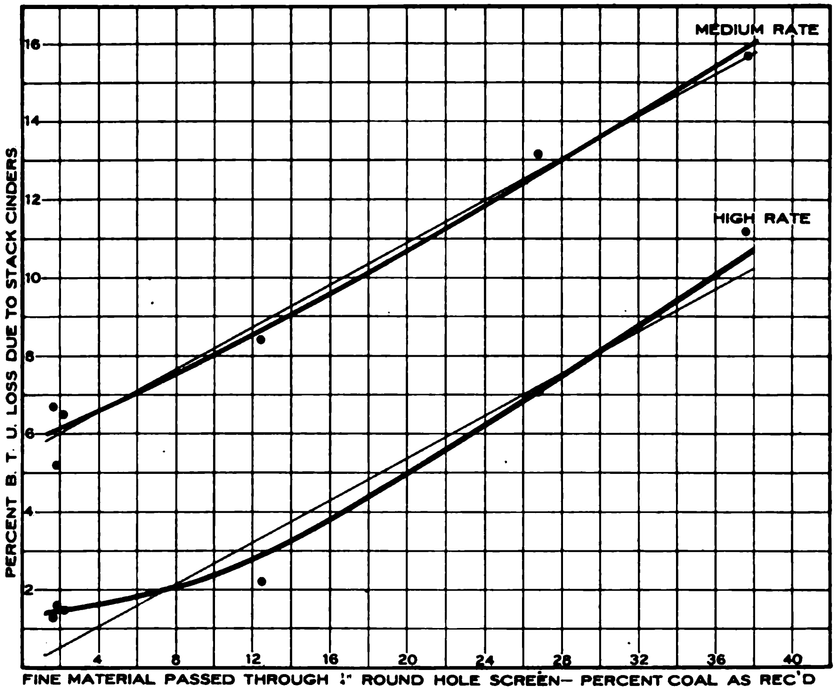


FIG. 20. THE RELATION BETWEEN CINDER LOSS AND THE PER CENT OF FINE MATERIAL IN THE COAL

corresponding loss for the 2-inch screenings was 13.2 per cent and for the 1¼-inch screenings 15.7 per cent. The average loss from the screenings during the high rate tests was more than twice as great as the average loss from the larger coals.

The figures and data indicate that with very fine coals such as screenings the cinder loss is large even at medium rates of combustion and with comparatively low front-end draft; but that under these conditions the cinder loss is not serious for the larger coals even when they contain a considerable amount of fine material as in mine run coal. For conditions involving high rates of combustion and strong drafts, however, the stack cinder loss is a serious one for all sizes of coal.

Fig. 20 shows the relation existing between the loss due to stack cinders and the amount of ¼-inch or smaller material in the coal as received. The data presented in Fig. 20 are also shown in Table 10. The curves in addition to showing the relative magnitude of the cinder losses for the two rates of operation, show that the cinder losses increased quite uniformly with the increase of fine material in the coal. At the medium rate about 1 per cent of the coal would apparently be lost as cinders if there were no ¼-inch fine material at all in the coal; while at the higher rate and without such material, the loss would be about 5.5 per cent. The curve for the high rate tests shows an increase in the cinder loss of very nearly one per cent for each increase of 3.7 per cent in the amount of ¼-inch material in the coal. The light straight lines in Fig. 20 show, for both rates, a uniform increase of one per cent in cinder loss for each 3.7 per cent increase in the ¼-inch material in the coal. The straight line represents the plotted points of the high rate tests closely but does not represent so well the points plotted for the medium rate tests. For the purposes of further discussion, however, the straight lines have been accepted as defining with sufficient accuracy the relations for both rates.

For conditions similar to those of the high rate tests, therefore, the percentage loss of fuel due to stack cinders may be expected to be approximately 5.5 plus the per cent of ¼-inch material in the coal divided by 3.7. Expressed as a formula this becomes

$$C = 5.5 + (F \div 3.7),$$

where C is the fuel loss on account of stack cinders expressed as per cent of B. t. u. in the coal, and F is the per cent of original coal passing

TABLE 10

PER CENT OF FINE MATERIAL IN COAL, AND LOSSES DUE TO STACK CINDERS

SIZE	Per Cent of Fine Material in Coal as Received, Passing Through			RATE	Loss Due to Stack Cinders		
					Per Cent of B.t.u. in Coal Fired	Wt. of Cinders in Per Cent of	
	¼" Round Hole Screen	½" Round Hole Screen	1" Round Hole Screen			Coal as Fired	Dry Coal
1	2	3	4	5	6	7	8
2" Lump..........	1.72	2.62	4.53	Medium.......	1.3	2.1	2.3
				High..........	6.7	7.4	8.4
2" x 3" Nut.......	1.87	2.93	5.77	Medium.......	1.6	2.3	2.5
				High..........	5.2	5.7	6.3
3" x 6" Egg.......	2.28	3.40	5.40	Medium.......	1.5	2.2	2.4
				High..........	6.5	7.2	7.9
Mine Run.........	12.50	19.94	31.30	Medium.......	2.2	3.1	3.3
				High..........	8.4	9.0	9.8
2" Screenings	26.88	41.09	66.82	Medium.......	7.1	8.9	9.8
				High..........	13.2	14.1	15.5
1¼" Screenings ...	37.59	57.62	95.56	Medium.......	11.2	12.5	13.6
				High..........	15.7	16.1	17.8

through a ¼-inch round hole screen. The expression for conditions similar to those of the medium rate tests is:

$$C = F \div 3.7.$$

The per cent of coal passing through a ¼-inch round hole screen has been used in the foregoing analysis since that was the smallest screen used in testing the coal for size. Similar analyses making use of the per cent of coal passing through a ½-inch or 1-inch screen show similar relations and result in similar formulas, with only a change in the value of the divisor. When F is the per cent of coal passing through a ½-inch round-hole screen, these formulas become:

$C = 5.5 + (F \div 5.7)$, for the high rate conditions,
$C = F \div 5.7$, for the medium rate conditions; and when F is the per cent of coal passing through a 1-inch round-hole screen, the corresponding formulas are:
$C = 5.5 + (F \div 9.3)$, for the high rate conditions,
$C = F \div 9.3$, for the medium rate conditions.

It should be remembered that kind of coal, intensity of draft, firebox and front end arrangement and probably other factors may materially affect the relations existing between cinder losses and the amount of fine material in coal, and that in the tests under consideration these variables have a very limited range. The results, therefore, if applied to conditions other than those from which they were derived should be used with caution and with an understanding of their limitations.

During tests with the four larger grades, larger quantities of cinders were collected than there was ¼-inch, or smaller, material in the coal. For these coals a considerable portion of the cinders must therefore have come from comparatively large pieces of coal. In the Screenings tests the cinders collected were materially less in amount than the ¼-inch or smaller material that existed in the coal. At all comparatively high rates of combustion therefore, and probably also at lower rates, there must be factors determining the amount of cinders produced other than the original amount of fine material in the coal fired.

19. *Heat Distribution.*—Fig. 21 presents average heat balances for the tests with each grade of coal for both medium and high rate tests. The figures have been so constructed that the groups are arranged with relation to decreasing values of the per cent of heat absorbed by the boiler during the high rate tests. This places the grades in the following order: nut, egg, mine run, lump, 2-inch screenings, and 1¼-inch screenings, the nut coal having shown the highest boiler efficiency, followed by the other sizes in the order named. During the medium rate tests the mine run coal showed the highest average boiler efficiency followed by egg, lump, nut, 2-inch screenings and 1¼-inch screenings in the order named. The percentages of heat absorbed by the boiler, during the medium rate tests, for the four grades of coal other than screenings, were, however, very nearly the same, ranging only from 75.8 per cent for mine run to 73.6 per cent for nut coal.

Fig. 21, in addition to the per cent of heat absorbed by the boiler, shows the following items, all in percentages of the heat of the coal fired; loss due to stack cinders; loss due to hydrogen in the coal, moisture in the coal, and moisture in the air; loss due to combustible in the ash; loss due to heat of the escaping gases; loss due to incomplete combustion of gases; and the "radiation and unaccounted for" loss. The complete heat balances, tabulated in Appendix III, present the same information in more detail and for each test. Fig. 21 reveals various relations concerning the heat distribution. For the high rate tests the figures representing cinder losses increase in size to about the same extent as the figures representing heat absorbed by the

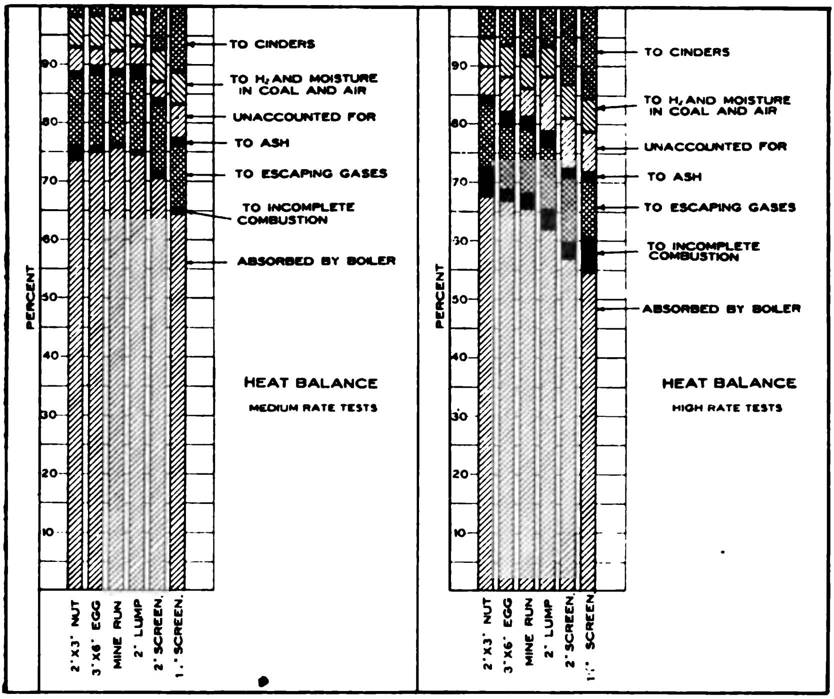


FIG. 21. THE DISTRIBUTION OF THE HEAT DURING BOTH THE MEDIUM RATE AND THE HIGH RATE TESTS

boiler decrease, in passing from nut to 1¼-inch screenings. In general also, all of the figures representing losses other than the cinder loss appear to be nearly equal. The principal exceptions to the general statements just made are found in the facts that losses due to incomplete combustion vary considerably, and that the heat distribution representing the lump coal tests shows a small cinder loss and a large "unaccounted for" loss. It may be said, however, that there was little variation in the losses due to escaping gases, to the ash-pan, to incomplete combustion, to moisture, and to radiation and unaccounted for, as between the different grades; and that the differences in the amounts of heat absorbed by the boiler are to be accounted for chiefly by the variation in the losses due to cinders.

This last statement is illustrated by Fig. 22, in which the height of each vertical band is proportioned to the sum of the heat absorbed by the boiler and the heat carried away in the cinders. It is obvious from the figure that, at the medium rate, the inferiority of the screenings as compared with the other sizes is entirely accounted for by their cinder losses. Among the four larger grades the cinder losses were small and nearly alike during the medium rate tests, and the differences in performances are not chargeable to cinders, but to other factors. Of these four sizes the mine run shows the highest boiler efficiency, despite the largest cinder loss. During the high rate tests

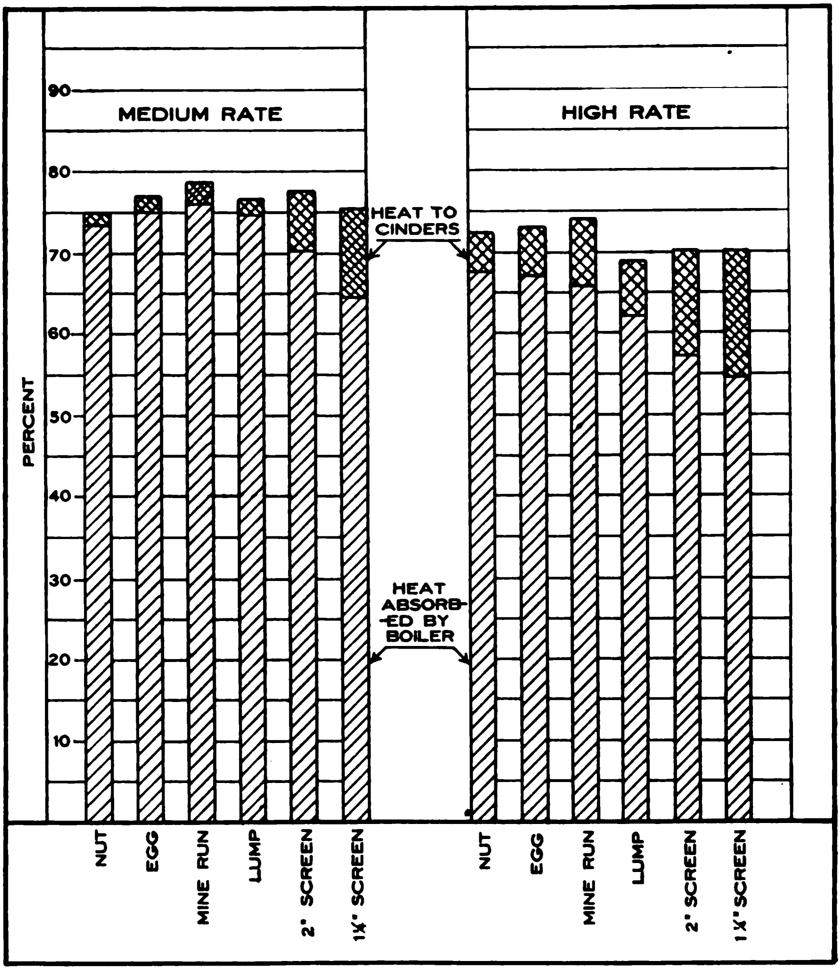


FIG. 22. THE SUM OF THE HEAT ABSORBED BY THE BOILER AND THE HEAT LOST IN THE CINDERS, FOR BOTH THE MEDIUM AND THE HIGH RATE TESTS

the inferiority of the screenings is again almost entirely accounted for by the cinder loss. The difference in performance of the four other grades is also apparently due chiefly to the heat carried away in the cinders, except as regards the lump coal which, although its cinder loss was less than that of the mine run and about equal to that of the egg and nut, nevertheless gave a performance inferior to all of them. This fact reflects the difficulty experienced in firing the lump coal at the high rate, which has been previously alluded to.

The radiation and unaccounted for losses are, for all grades, quite uniform, the minimum and maximum values for the twelve groups being 3.0 per cent and 9.3 per cent. The minimum, maximum, and average values for the entire 36 tests are 2, 11.2, and 5.2 per cent respectively. If there were no unaccounted for loss the average value of 5.2 per cent should represent with some degree of exactness the radiation loss. In addition to the heat losses accounted for there are probably other losses not measured, such as those due to sensible heat carried away by the ash and cinders, unburned combustible gases not determined by the gas analysis, and unburned carbon in the smoke other than the cinders which are collected. Making an allowance of 1 or 2 per cent for the losses just mentioned and deducting this from the total average unaccounted for—5.2 per cent—would leave the average value for the loss due to radiation at about 3 or 4 per cent. While we have no very reliable data as to the radiation loss under conditions similar to those of the tests, the figure 3 or 4 per cent is probably not greatly in error. There is, further, for all tests a comparatively small variation of the radiation and unaccounted for loss from the mean value. Because of these facts it is fair to conclude that, in general, the heat distributions as given in the tables account for practically all of the heat content of the coal; that the amounts actually unaccounted for are so small as not seriously to invalidate any portion of the balances; and finally that the approximately complete and correct accounting for of all the heat content of the coal makes it probable that values defining the heat distribution may safely be taken as a basis for conclusions concerning the test results.

IX. CONCLUSIONS

Such generalizations as follow seem warranted by the test results. They are presented as applicable only to the coal tested. How closely they apply to coals from other fields is not clear, although it is probable that they hold good for other coals of like mechanical make-up and similar physical properties. If it is desired to apply these conclusions to coals from other fields, the facts should be borne in mind that the six sizes tested were more nearly alike in chemical composition and heating value than is often the case, that the cinder losses account in large measure for the differences in performance, that the firing was unusually uniform and constantly supervised, that the large lumps in both the mine run and lump coals were broken before firing, and that the same exhaust nozzle was used throughout all tests.

The purpose of the tests and the general program are set forth in Chapter II.

The heating values and the chemical analyses of the six sizes of coal tested are given in Table 1 of Chapter III, and their mechanical make-up is defined in sections 6 and 7 of that chapter.

The final results of the tests, expressed in terms of equivalent evaporation per pound of dry coal, are presented in Columns 2 and 4 of Table 9 in Chapter VIII, and they are discussed at the end of section 17 in that chapter.

The relative values of the six sizes are defined by the percentage values given in Columns 3 and 5 of Table 9 in Chapter VIII, and are illustrated by Figs. 17 and 18. It should not be forgotten that these percentages define the relative values of the coals on the tender —not at the mine.

At the prices which prevailed when the tests were made, both sizes of screenings were slightly more economical than the mine run coal. Among the four larger grades the mine run was much more economical than either the egg, nut, or lump coals. Averaging the results at both rates of evaporation, the price differential between 2-inch screenings and 1¼-inch screenings was just offset by the superior performance of the former.

Except as regards the lump coal at the high rate of evaporation and the four larger grades at the medium rate, the heat lost in the cinders accounts almost entirely for the differences in performance among the various grades. These losses are shown in Figs. 21 and 22 and they are discussed in sections 18 and 19. For the Screenings they varied during the medium rate tests from 7.1 to 11.2 per cent, and in the high rate tests from 13.2 to 15.7 per cent. Among the four larger sizes the heat lost in the cinders varied during the medium rate tests from 1.3 to 2.2 per cent, and in the high rate tests from 5.2 to 8.4 per cent.

Inspection of Figs. 21 and 22 reveals the fact that, despite greater heat loss in the cinders, mine run coal at the medium rate of evaporation had a higher boiler efficiency than either the egg or the lump; and at the high rate its efficiency was greater than that of the lump, and only 1.3 per cent inferior to that of the egg. It is assumed that this is due to the better combustion of the smaller pieces of coal, which are more numerous in the mine run than in the two other sizes.

The inferiority of the performance of the nut coal at the medium rate was probably due to insufficient draft. Its superior performance at the high rate is considered to be due to its small cinder loss and to the evenness and uniformity of the fire which it was possible to maintain with this grade.

At the high rate of evaporation it was more difficult to handle the fire with lump coal than with mine run; and at both rates the evaporative efficiency of the lump was less than that of mine run. The test results offer, therefore, no support for the popular belief in the superiority of lump coal.

As stated in Chapter III the large lumps in both the mine run and lump coals were broken before firing—the former somewhat the more thoroughly, as is evidenced by the fact that after being thus cracked all of the mine run would pass a 5-inch round opening, whereas only 74 per cent of the lump would pass an opening of this size. As has been stated, the evaporative efficiency of the mine run was greater than that of the lump at both rates of evaporation. Since these two coals were not in other respects identical, the facts cited do not form a conclusive argument for the advantage of breaking the large lumps; but, taken in connection with the firing experience in the laboratory, they do offer support for the opinion expressed by the Fuel Test Committee that the cracking of coal to the point where it will all pass a 5-inch or 6-inch round-hole screen is worth more than it costs at well equipped coal chutes.

APPENDIX I

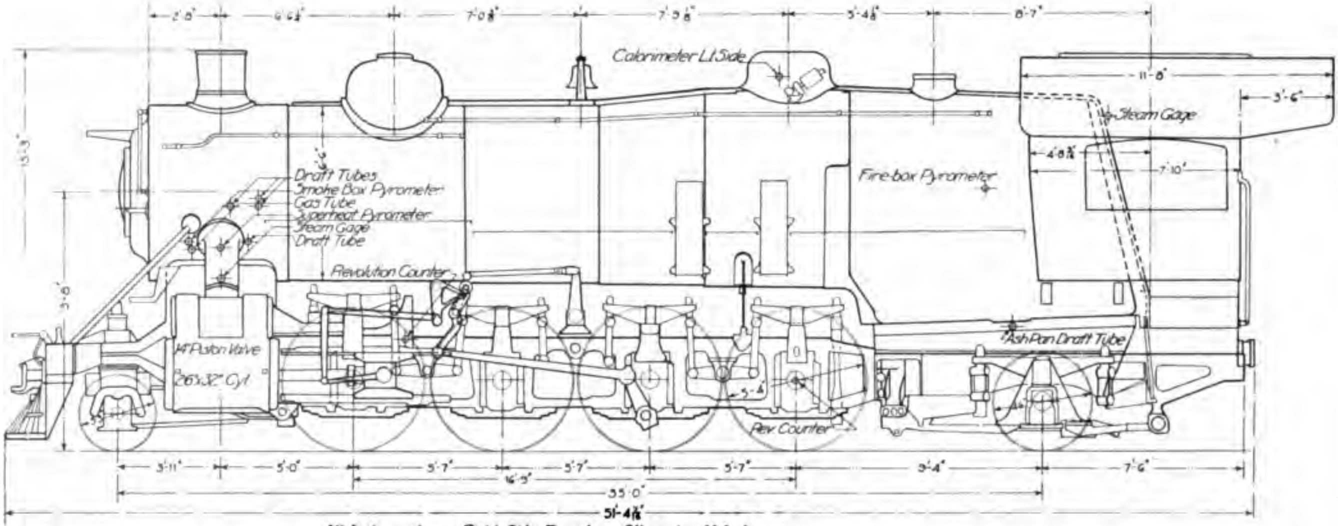
THE LOCOMOTIVE

The locomotive has been briefly described in the body of this report. For convenience of reference some of the facts there cited are repeated in this appendix which is intended to describe the locomotive in detail.

20. *General Design.*—Baltimore and Ohio locomotive 4846 is of the 2-8-2 type and is shown in general design in Figs. 12, 23, 24, and 25. It was built by the Baldwin Locomotive Works in the summer of 1916. It uses superheated steam at 190-pound boiler pressure, in simple cylinders, 26 inches in diameter by 32 inches stroke. Its principal general dimensions are as follows:—

Weight of locomotive, in working order	284500 lb.
Weight of tender, loaded	180000 lb.
Weight of locomotive and tender, in working order . .	464500 lb.
Weight on front drivers	55600 lb.
Weight on intermediate drivers	54900 lb.
Weight on main drivers	56200 lb.
Weight on back drivers	55300 lb.
Weight on drivers, total	222000 lb.
Weight on leading truck	19400 lb.
Weight on trailing truck	43100 lb.
Nominal maximum tractive effort	54587 lb.
Driving wheel base	16 ft.- 9 in.
Total wheel base of locomotive	35 ft.- 0 in.
Driving wheel diameter—nominal	64 in.
Driving wheel diameter—actual	63.92 in.
Leading truck wheel diameter	33 in.
Trailing truck wheel diameter	46 in.
Main driving journals	11½ x 21 in.
Other driving journals	9½ x 13 in.
Leading truck journals	6 x 6 in.
Trailing truck journals	8 x 14 in.

21. *The Boiler, Firebox, and Front End.*—The boiler, the general design of which is shown in Figs. 26 and 27, was of the wagon top radial stay type, composed of four ring courses and the back end. The main steam dome was mounted over an opening about 27 inches in diameter and an auxiliary dome was mounted on the back end, about one-third of the length of the firebox back of the flue sheet. Flexible staybolts were used throughout.



All Instruments on Right Side Except as Otherwise Noted

FIG. 23. SIDE ELEVATION OF BALTIMORE AND OHIO RAILROAD LOCOMOTIVE, 4846.

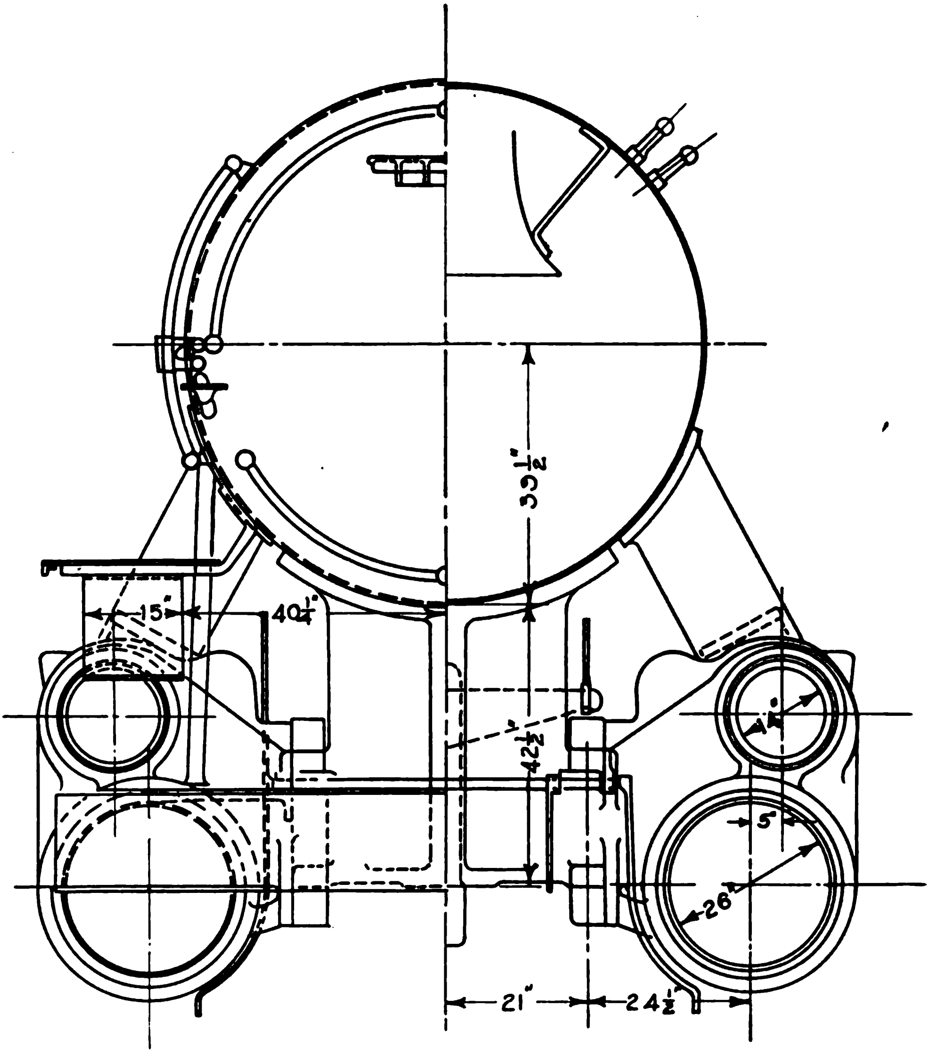


FIG. 24. PARTIAL FRONT ELEVATION

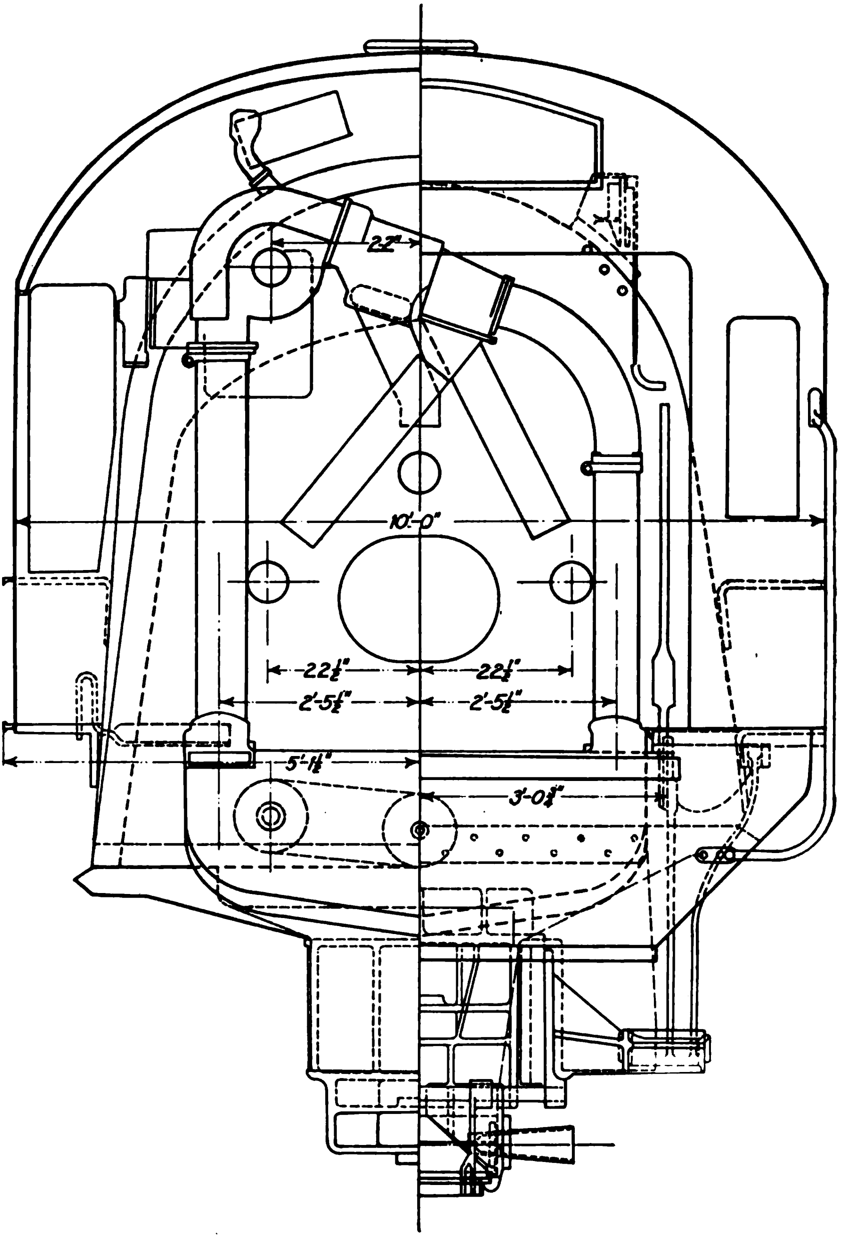


Fig. 25. Rear Elevation and Section Through the Cab

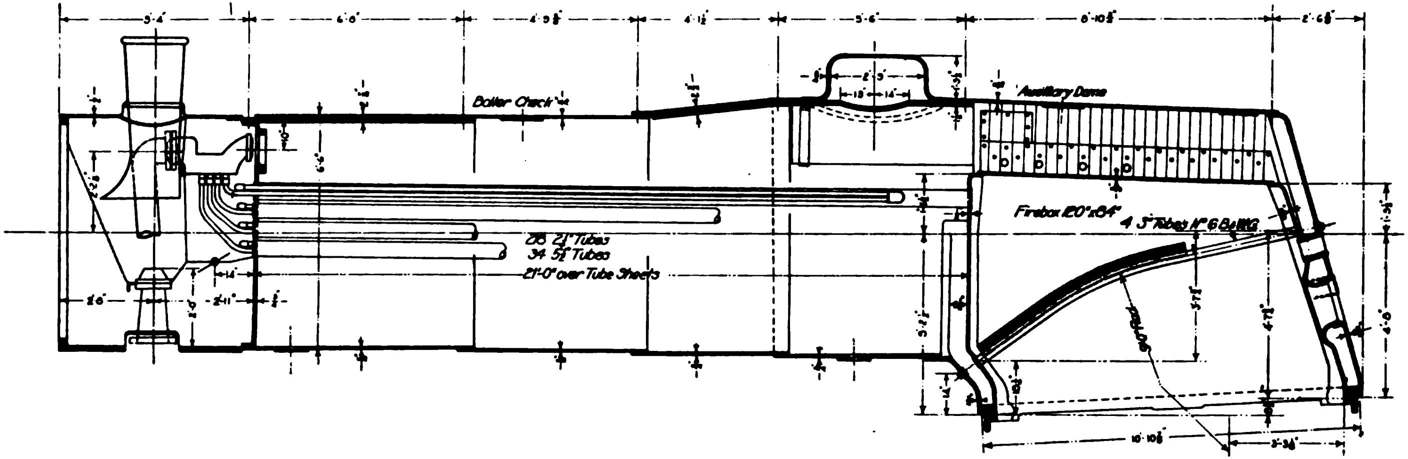


Fig. 26. Longitudinal Section Through the Boiler

The firebox was provided with a "Security" brick arch carried on four 3-inch arch tubes. The grates—shown in Fig. 28—were of the box type with a total area of 69.8 square feet and a total area through the grate openings of 17.0 square feet—24.4 per cent of the grate area. In ordinary operation the firebox was fed by a Street mechanical stoker built by The Locomotive Stoker Company. Its general design is shown in Fig. 25. Three inlets of 5½-inch inside diameter were provided in the back head for the stoker nozzles.

The general design of the front end and the superheater appears in Figs. 26 and 27. While the locomotive was being broken in in service the front-end arrangements shown in the figures were tested by using coal similar to that to be used during the tests and were found to be satisfactory. They were not modified during the progress of the tests. The superheater was of the Schmidt top-header type and consisted of 34 elements. The principal boiler dimensions appear in the following list:

Outside diameter of first ring	78 in.
Cylindrical courses, thickness of sheet	11/16 and ¾ in.
Wrapper sheet, thickness	9/16 in.
Back flue sheet, thickness	½ in.
Front flue sheet, thickness	¾ in.
Firebox sheets, thickness	⅜ in.
Number of 2¼-inch tubes	218
Number of 5½-inch tubes	34
Number of 3-inch arch tubes	4
Length between tube sheets	21 ft.
Water space in the boiler	547.3 cu. ft.
Steam space in the boiler	144.7 cu. ft.
Heating surface of 2¼-inch tubes, fireside	2,410.0 sq. ft.
Heating surface of 5½-inch tubes, fireside	972.8 sq. ft.
Heating surface of 3-inch tubes, fireside	31.4 sq. ft.
Heating surface of front tube sheet, fireside	15.3 sq. ft.
Heating surface of firebox, fireside	200.4 sq. ft.
Total water heating surface, fireside	3,630.0 sq. ft.
Superheating surface, fireside	1,030.0 sq. ft.
Total water and superheating surface, fireside	4,660.0 sq. ft.
Number of superheater tubes	136
Outside diameter of superheater tubes	1 7/16 in.
Total length of superheater tubes	2,733.5 ft.
Length of firebox, inside	120 in.
Width of firebox, inside	84 in.
Depth of firebox, at front	81 in.
Depth of firebox, at back	71½ in.
Volume of firebox	348.6 cu. ft.
Grate area	69.8 sq. ft.
Exhaust nozzle, tip diameter	6 in.

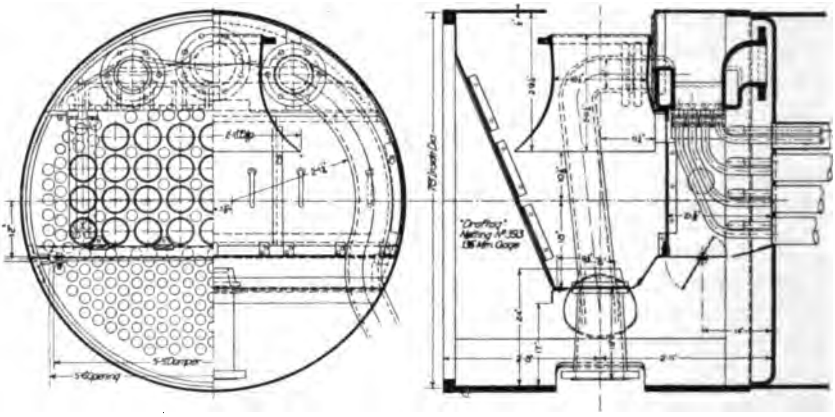


FIG. 27. THE FRONT-END ARRANGEMENT AND THE SUPERHEATER

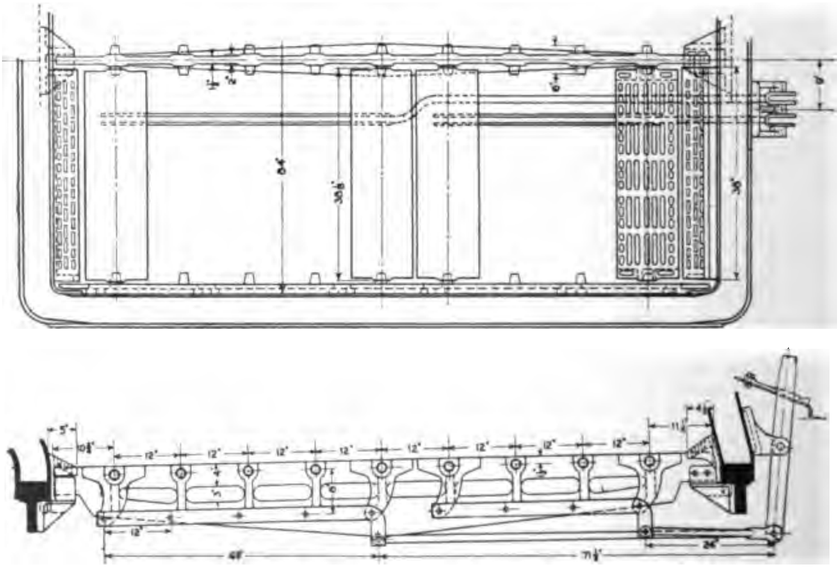


FIG. 28. THE GRATES

22. *The Cylinders and the Valves.*—The arrangement of the cylinders and the valves is shown in Fig. 24. The valves were driven by a Baker-Pilliod gear. The following list presents the principal cylinder and valve dimensions together with data useful in interpreting the indicator diagrams:

Cylinder diameter, right side	25.771 in.
Cylinder diameter, left side	25.767 in.
Valve chamber diameter, right side	14.0 in.
Valve chamber diameter, left side	14.0 in.
Stroke of piston, both sides	32.0 in.
Piston rod diameter, both sides	4.0 in.
Piston displacements:	
Right side, head end	9.660 cu. ft.
Right side, crank end	9.427 cu. ft.
Left side, head end	9.657 cu. ft.
Left side, crank end	9.424 cu. ft.

Clearance volumes—per cent of piston displacement:

Right side, head end	11.0 per cent
Right side, crank end	11.5 per cent
Left side, head end	11.1 per cent
Left side, crank end	11.4 per cent

APPENDIX II

TEST METHODS AND CALCULATIONS

The test methods employed were, in general, those outlined in the "Method of Conducting Locomotive and Road Tests" as published in the Proceedings of the American Railway Master Mechanics' Association, Volume 47, page 538.

All tests were run under one of two sets of conditions as to speed and cut-off: The "Medium Rate" tests at a speed of 100 revolutions per minute and at 33 per cent cut-off, and the "High Rate" test at a speed of 135 revolutions per minute and at 55 per cent cut-off. The test methods employed were the same for all tests, and throughout each test all conditions subject to control were maintained as nearly constant as possible. The graphical logs in Appendix III show to what extent uniformity of test conditions was obtained during tests 2416 and 2405, and these logs may be taken as fairly representative of test conditions for all of the tests.

All instruments were known to be correct within reasonable limits or were calibrated at intervals and suitable corrections applied to the observed data. Observations were in general taken every ten minutes. Indicator diagrams were taken from each end of both cylinders at intervals varying from ten minutes on some tests to forty minutes on other tests. Owing to the uniformity of test conditions and to the fact that only two sets of conditions as to speed and cut-off were employed, the taking of indicator diagrams more frequently was unnecessary. The locations of the more important instruments and apparatus are indicated in the figures in Appendix I.

23. *Duration of Tests.*—The tests varied in length from 58 minutes for test 2435 to 6 hours and 17 minutes for test 2401. The general test program contemplated one medium and one high rate test for each size of coal during which approximately ten tons of coal should be burned per test; and two medium and two high rate tests for each grade of coal during which approximately 6⅔ tons of coal should be burned per test. An examination of the data shows that during five tests ten tons or more of coal were burned per test; that during 30 tests the amount of coal per test varied from 4 to 9 tons; and that in one test only 2 tons of coal were burned. For the entire 36 tests the average amount of coal burned was 7 tons per test. As an average therefore about 200 pounds of coal per square foot of grate were burned per test.

24. *Beginning and Closing a Test.*—Fires were built upon a clean grate for each test. With sufficient steam pressure, the locomotive was started and was gradually brought to the required conditions of speed and cut-off. It was then operated for a short time under the required conditions and until a satisfactory fire and a satisfactory boiler pressure were being maintained. On signal the ash pan and cinder separator were closed, observations of water levels and steam pressure were made, and the test was begun. In closing a test simultaneous observations were made upon water levels, steam pressure and condition of fire. The locomotive was then stopped as quickly as conditions warranted. As soon as possible after stopping, ashes were removed from ash pan, and cinders from the cinder separator.

In all cases it was endeavored to have the same amount of combustible matter upon the grate at the close as at the start of the test. The removal of ash from the fire in connection with the closing of the test was primarily for the purpose of judging the amount of combustible upon the grate and not for the purpose of collecting ash. The endeavor was made to have the boiler pressure and the water level in the boiler substantially the same at the close as at the beginning of the test. Corrections were made for such irregularities as occurred.

25. *Temperatures, Pressures, etc.*—Temperatures in the firebox were measured by a platinum and platinum-rhodium thermocouple; and front-end and superheated steam temperatures by copper and copper-constantan couples. Mercury thermometers were used at other points where temperature observations were made.

Boiler pressure observations were taken from a gauge located in the engine cab. Draft pressures were measured by means of "U" tubes with water or with differential draft gauges. Quality of steam was determined by means of a throttling calorimeter fitted with a suitable sampling tube. During portions of a few tests the moisture in the steam was so great that it could not be measured by means of the throttling calorimeter. Speed was measured by means of a stroke counter.

26. *Flue Gas Sampling and Analysis.*—Front-end gas samples were collected through a sampling pipe provided with numerous small holes along the pipe through which the gas was drawn. The time during which a single sample was collected varied from 20 to 60 minutes, depending mainly upon the total length of the test. The taking of samples covered in general the entire time of the test. All samples were collected over mercury and analyzed immediately after collection. The apparatus used for the analysis of the flue gases was

Burrell and Seibert's modification of Haldane's apparatus. The accuracy of this apparatus is sufficient to distinguish 0.01 per cent of carbon monoxide, of methane, or of hydrogen. In the present work, CO_2 percentages were checked to 0.02 per cent and unusual care was taken both in the collection of samples and in the analysis in order that reliable data might be secured regarding the percentages of carbon monoxide, of methane, and of hydrogen.

The tabulated data relating to the composition of the flue gases, as well as the heat losses due to methane and to hydrogen, indicate that under ordinary conditions very little of the original heat of the coal is lost because of the presence of these gases and that only a small error will be made if the volume of these gases which is present be treated as carbon monoxide instead of as methane and hydrogen.

27. *Samples of Coal, Ash, and Cinders for Chemical Analysis.*—Following the close of a test, the ashes collected in the ash pan and the cinders collected in the cinder separator were weighed and sampled. Samples weighing from 50 to 150 pounds were collected as the ash and cinders were being weighed, a small amount being taken from each barrow load after passing over the scales.

Ninety-five per cent or more of each cinder sample being smaller than $\frac{1}{16}$ inch, the large sample was thoroughly mixed and reduced by "quartering" to a five-pound sample. The ashes were mixed and crushed to ¼-inch size and reduced to a five-pound sample by "quartering."

The general practice of sampling the coal for chemical analysis was that outlined in the 1915 Year Book of the American Society for Testing Materials. During each test, as the coal was loaded from the bins into the wagons to be transferred to the firing platform, amounts weighing approximately 15 pounds (one scoopful) were placed in sampling cans. The number of these portions was so proportioned that a total sample of 1,000 pounds would be collected from the total amount of coal fired during one test. In the case of the 1¼-inch and the 2-inch screenings, because of their general uniformity and thorough mixture resulting from the process of screening and loading, the number of scoops of sample coal was so proportioned to the gross amount of coal burned that total samples weighing approximately 500 pounds instead of 1,000 pounds were collected. For test 2435 a sample of only 200 pounds was collected. For all other tests the samples weighed 500 pounds or more. The average weight of all samples collected for the grades larger than screenings was 885 pounds.

Special care was exercised to ensure that the coal selected for samples was in all respects representative of the coal being fired. In general, samples for the chemical laboratory were prepared from the large samples immediately after collection. The samples were prepared largely by mechanical means which produced results equivalent to the hand method described in the year book of the American Society for Testing Materials. The entire sample was crushed by rolls to less than 1-inch size, then mixed by "coning" and reduced by "long pile" mixing and "quartering" to from 125 to 250 pounds. This amount was then pulverized and, through quartering, was reduced to a five-pound sample.

The five-pound samples of coal, ash, and cinders were submitted to the chemical laboratory for analysis.

28. *Chemical Analysis of Coal, Ash, and Cinders.*—The chemical analysis and heat determinations were made by the United States Bureau of Mines at the Experiment Station Laboratory, Pittsburgh, Pa. The methods of analysis and details of the apparatus used by the Bureau of Mines in analyzing coal are fully described in Technical Paper 8 issued by the bureau in June, 1913, and all samples of coal, ash and stack cinders were analyzed in accordance with those methods.

Proximate analyses and direct B. t. u. determinations were made for the coal sample for each test. One ultimate analysis was made for each size of coal tested. The ultimate analyses were made from composite samples. Each composite sample was made by combining equal parts by weight from the air-dried samples representing the tests for each grade of coal. The ultimate analyses for the individual tests which appear in the report are based upon the percentages of moisture, ash, and sulphur as determined by the proximate analysis; and upon the assumption that the percentages of carbon, hydrogen, oxygen, and nitrogen as determined for the individual tests are proportional to the percentages of carbon, hydrogen, oxygen, and nitrogen as determined for the composite samples of that size by ultimate analysis.

Proximate analyses and direct B. t. u. determinations of the cinder samples were made for each test. Proximate analyses were made of the ash sample for each test and direct B. t. u. determinations were made for each ash sample for tests 2400 to 2427 inclusive. For the ash samples subsequent to test 2427, the B. t. u. values were calculated from an average B. t. u. value for one pound of moisture-free and ash-free content of the ash samples. The average moisture-free and ash-free B. t. u. value for all ash samples 2400 to 2424 inclusive, is 14,148 B. t. u. per pound, and all ash samples subsequent to test 2427 have B. t. u. values dependent upon this average value and proportional to the moisture-free and ash-free content of the individual samples.

29. *Samples of Coal for Mechanical Analysis.*—From each car of coal delivered, a sample was taken for mechanical analysis to determine the grade percentages in each size of coal. All samples were collected in uniform manner, the handling from car to separating screens being such that approximately the same amount of incidental breakage took place as occurred when the regular firing coal was transferred from the cars to the firing platform. As each car of the run of mine and the 2-inch lump coal was unloaded, every twentieth scoopful and every twentieth lump unloaded by hand were set aside. In the case of the other coals, which contained no large lumps, every fifteenth scoopful was set aside. The weight of each sample collected was about five per cent of the weight of the coal in each car.

30. *Smoke Records.*—The Ringelmann scale was used in making the smoke observations. Nos. 1, 2, 3, 4, and 5 of the Ringelmann chart represent respectively, 20, 40, 60, 80 and 100 per cent of black smoke. Owing to the large amount of steam escaping with the stack gases, changes in temperature and light greatly affect the appearance of the smoke as regards its apparent blackness. Due to these and other causes which affect the value of observations of this kind, the tabulated results regarding blackness of smoke should be accepted as only approximately correct.

31. *Methods of Calculation.*—The methods used in determining the calculated results are in general similar to the detailed methods of calculation published in Bulletin No. 82, University of Illinois, Engineering Experiment Station.

The calculations relating to heat losses due to the presence of hydrogen and methane in the escaping gases were based upon the determination of the amounts of these gases present and upon heat values of 62100 and 23842 B. t. u. per pound for hydrogen and methane, respectively.

The steam tables of G. A. Goodenough, presented in "Properties of Steam and Ammonia," have been used in all calculations pertaining to the properties of steam.

Certain methods of calculation relating to the determination of the amount of superheated steam produced and the amount used by the engine are as follows:

Item 409. Degrees of Superheat
(Branch-pipe Temperature)—
(Temperature of Saturated Steam at Branch-pipe pressure).

Item 644. Factor of Evaporation

$$\frac{Hs - h}{971.7}$$

H^s = Total heat of steam at branch-pipe pressure.
h = Heat of liquid due to feed water temperature.

Heat Transfer Across Water Heating Surface per Minute, B. t. u.

Item 633 $\times (q + xr - h) \div 60$

$q + xr - h$ = the heat added to each pound of water evaporated by the boiler exclusive of the superheater.

Heat Transfer Across Superheater Heating Surface per Minute, B. t. u.

(Pounds of steam to superheater per minute) $\times$ $(Hs - q - xr)$

$Hs - q - xr$ = the heat added to each pound of steam passing through the superheater.

Item 645. Equivalent Evaporation per Hour, Pounds.

$$\left[\begin{array}{l}\text{(Heat transfer per hour across water HS)} + \\ \text{(Heat transfer per hour across Superheater } HS\text{)}\end{array}\right] \div 971.7$$

Superheated Steam per Hours, Pounds.

Item 645 ÷ Item 644

Superheated Steam to Engine per Hour, Pounds.
(Superheated steam per hour) —
(Superheated steam loss per hour due to Calorimeter leaks, Corrections, etc.)

APPENDIX III

Tabulated Data and Results

The purpose of this appendix is to present, for the sake of those who are interested in the details of the tests, all the data and the results. The appendix consists of sixteen tables and two figures.

Tables 11 to 26, inclusive, contain the results for each of the 36 tests arranged in six groups. Each of the six groups presents the data and the results for a particular size of fuel. Within each group the arrangement is such that the medium rate tests precede the high rate tests. The tests were numbered consecutively in the order in which they were run and their arrangement within the tables is, with few exceptions, also in this order. Under each size of fuel the results of all tests made at a common rate of evaporation have been averaged and these averages appear in the tables in bold face type. The columns headed "Test Number" and "Laboratory Designation" are repeated from table to table to facilitate cross reference. The first term of the column headed "Laboratory Designation" indicates the kind of fuel; the second, the nominal speed in revolutions per minute; and the third, the nominal cut-off in per cent of stroke. The abbreviations used in this column are: M. R. for mine run; 2 in. S. for 2-inch screenings; and 1¼ in. S. for 1¼-inch screenings. The data and the results are presented under 146 column headings. The numbers assigned to these columns are included between 344 and 900 and they appear in the tables in the order of these numbers, which are in general the same as those used in the code for testing locomotives published in the Proceedings of the American Railway Master Mechanics' Association, Vol. 47, p. 538.

In Fig. 29 and Fig. 30 are shown graphical logs for tests 2416 and 2405 respectively, which are fairly typical of all the tests. Test 2416 is a medium rate test, during which 19915 pounds of 3-inch by 6-inch egg coal were fired; whereas No. 2405 is a high rate test during which 20,000 pounds of mine run coal were fired. The lines plotted in these two figures afford a basis for judging of the uniformity of the test conditions which prevailed during these tests.

TABLE 11

GENERAL CONDITIONS

TEST NUMBER	Laboratory Designation	Duration of Test, Hours	SPEED				Reverse Lever Notches from Center	Throttle
			Revolutions		Equivalent			
			Total	Average per Minute	Speed in Miles per Hour	Piston Speed in Feet per Minute		
	Code Item ☞	345	351	352	353	354	360	363
2400	M. R.-100-33	3.62	21576	99.4	18.9	530.1	2	Full
2401	M. R.-100-33	6.28	37439	99.3	18.9	529.6	2	Full
2402	M. R.-100-33	5.20	31101	99.7	19.0	531.7	2	Full
	Average	....	....	**99.5**	**18.9**	**530.5**	....	....
2405	M. R.-135-55	2.72	21807	133.8	25.5	713.6	6	Full
2406	M. R.-135-55	2.68	21687	134.7	25.6	718.4	6	Full
2429	M. R.-135-55	1.92	15529	135.0	25.7	720.0	6	Full
	Average	....	....	**134.5**	**25.6**	**717.3**	....	....
2408	Nut-100-33	3.77	22528	99.7	19.0	531.7	2	Full
2409	Nut-100-33	2.33	13904	99.3	18.9	529.6	2	Full
2410	Nut-100-33	4.33	25886	99.6	19.0	531.2	2	Full
2426	Nut-100-33	4.50	26782	99.2	18.9	529.0	2	Full
	Average	....	....	**99.5**	**19.0**	**530.4**	....	....
2412	Nut-135-55	2.67	21643	135.3	25.8	721.6	6	Full
2413	Nut-135-55	3.00	24285	134.9	25.7	719.4	6	Full
2414	Nut-135-55	2.00	16189	134.9	25.7	719.4	6	Full
	Average	....	....	**135.0**	**25.7**	**720.1**	....	....
2415	Egg-100-33	3.50	20834	99.2	18.9	529.0	2	Full
2416	Egg-100-33	5.83	34822	99.5	18.9	530.6	2	Full
2423	Egg-100-33	4.00	23741	98.9	18.8	527.4	2	Full
	Average	....	....	**99.2**	**18.9**	**529.0**	....	....
2420	Egg-135-55	2.00	16214	135.1	25.7	720.5	6	Full
2422	Egg-135-55	2.17	17650	135.8	25.9	724.2	6	Full
2424	Egg-135-55	1.98	16133	135.6	25.8	723.2	6	Full
	Average	....	....	**135.5**	**25.8**	**722.6**	....	....
2417	Lump-100-33	4.00	23837	99.3	18.9	529.6	2	Full
2418	Lump-100-33	5.83	34894	99.7	19.0	531.7	2	Full
2419	Lump-100-33	3.67	21918	99.6	19.0	531.2	2	Full
	Average	....	....	**99.5**	**19.0**	**530.8**	....	....
2425	Lump-135-55	1.00	8111	135.2	25.7	721.0	6	Full
2427	Lump-135-55	1.50	12194	135.5	25.8	722.7	6	Full
2428	Lump-135-55	1.83	14963	136.0	25.9	725.3	6	Full
2442	Lump-135-55	2.00	16071	133.9	25.5	714.1	6	Full
	Average	....	....	**135.2**	**25.7**	**720.8**	....	....
2430	2 in. S.-100-33	2.62	15671	99.8	19.0	532.2	2	Full
2434	2 in. S.-100-33	3.13	18475	98.3	18.7	524.2	2	Full
2435	2 in. S.-100-33	0.97	5791	99.8	19.0	532.2	2	Full
	Average	....	....	**99.3**	**18.9**	**529.5**	....	....
2436	2 in. S.-135-55	1.35	10870	134.2	25.5	715.7	6	Full
2437	2 in. S.-135-55	1.50	12133	134.8	25.7	718.9	6	Full
	Average	....	....	**134.5**	**25.6**	**717.3**	....	....
2431	1¼ in. S.-100-33	1.77	10603	100.0	19.0	533.3	2	Full
2432	1¼ in. S.-100-33	1.87	11184	99.9	19.0	532.8	2	Full
2433	1¼ in. S.-100-33	3.10	18511	99.5	18.9	530.6	2	Full
	Average	....	....	**99.8**	**19.0**	**532.2**	....	....
2440	1¼ in. S.-135-55	1.50	12063	134.0	25.5	714.6	6	Full
2441	1¼ in. S.-135-55	1.50	12105	134.5	25.6	717.3	6	Full
	Average	....	....	**134.3**	**25.6**	**716.0**	....	....

TABLE 12

TEMPERATURES

TEST NUMBER	Laboratory Designation	TEMPERATURE, DEGREES F.						
		Front-End	Laboratory		Branch Pipe	Feed Water	Fire-Box	Out-Door
			Dry Bulb	Wet Bulb				
	Code Item ☞	**367**	**363**	**369**	**370**	**373**	**374**	....
2400	M. R. -100-33	535	75	71	573	58.9	1735	51
2401	M. R. -100-33	535	73	70	566	56.0	1835	49
2402	M. R. -100-33	539	79	78	564	59.2	1812	62
	Average	**536**	**76**	....	**568**	**58.0**	**1794**	....
2405	M. R.-135-55	627	64	61	628	56.5	2271	35
2406	M. R.-135-55	631	60	58	631	55.6	2334	25
2429	M. R.-135-55	624	48	44	618	55.2	2140	37
	Average	**627**	**57**	....	**626**	**55.8**	**2248**	....
2408	Nut-100-33	595	51	54	580	55.2	2090	12
2409	Nut-100-33	588	54	55	582	55.2	2034	10
2410	Nut-100-33	570	58	56	569	54.1	2008	12
2426	Nut-100-33	555	49	48	572	54.9	1967	23
	Average	**577**	**53**	....	**578**	**54.9**	**2025**	....
2412	Nut-135-55	607	50	55	629	55.7	2293	28
2413	Nut-135-55	611	45	52	632	54.5	2267	18
2414	Nut-135-55	631	51	56	634	55.9	2174	28
	Average	**616**	**49**	....	**632**	**55.4**	**2245**	....
2415	Egg-100-33	543	65	64	576	55.9	1808	44
2416	Egg-100-33	540	62	63	574	56.2	1801	42
2423	Egg-100-33	539	46	47	571	54.8	....	8
	Average	**541**	**58**	....	**574**	**55.6**	**1805**	....
2420	Egg-135-55	588	58	56	610	55.9	2210	42
2422	Egg-135-55	634	56	57	590	54.7	2278	35
2424	Egg-135-55	626	51	53	617	55.0	2183	9
	Average	**616**	**55**	....	**606**	**55.2**	**2224**	....
2417	Lump-100-33	546	59	58	578	56.0	1838	46
2418	Lump-100-33	545	58	58	578	57.0	1857	36
2419	Lump-100-33	553	57	57	578	56.2	1849	36
	Average	**548**	**58**	....	**578**	**56.4**	**1848**	....
2425	Lump-135-55	618	46	47	595	56.3	2178	24
2427	Lump-135-55	625	50	46	578	55.3	2308	26
2428	Lump-135-55	635	43	42	603	54.4	2277	24
2442	Lump-135-55	637	49	42	616	54.4	2192	18
	Average	**629**	**47**	....	**598**	**55.1**	**2239**	....
2430	2 in. S.-100-33	549	56	48	583	56.4	2010	29
2434	2 in. S.-100-33	541	48	46	584	54.8	1817	—2
2435	2 in. S.-100-33	549	62	49	578	57.1	1936	25
	Average	**546**	**55**	....	**582**	**56.1**	**1921**	....
2436	2 in. S.-135-55	631	40	38	634	53.4	2078	5
2437	2 in. S.-135-55	634	44	40	637	54.1	2194	27
	Average	**633**	**42**	....	**636**	**53.8**	**2136**	....
2431	1¼ in. S.-100-33	551	63	59	589	57.0	2003	48
2432	1¼ in. S.-100-33	544	76	66	591	57.6	1798	60
2433	1¼ in. S.-100-33	543	59	54	572	55.3	1874	13
	Average	**546**	**66**	....	**584**	**56.6**	**1892**	....
2440	1¼ in. S.-135-55	634	42	38	604	54.1	2273	9
2441	1¼ in. S.-135-55	639	43	38	629	54.0	2234	10
	Average	**637**	**43**	....	**617**	**54.1**	**2254**	....

TABLE 13

PRESSURES

TEST NUMBER	Laboratory Designation	Pressure—lb. per sq. in.			Draft, in. of Water				
		Boiler, Average Gauge	Branch Pipe Average Gauge	Laboratory Barometric	Front End			Fire Box	Ash Pan
					Front of Diaphragm	Back of Diaphragm			
						Below Damper	Above Damper		
	Code Item☞	380	383	383	394	395	....	396	397
2400	M. R.-100-33	190.4	179	14.3	2.8	2.3	2.1	1.2	0.2
2401	M. R.-100-33	190.0	172	14.3	2.8	2.4	2.1	1.5	0.2
2402	M. R.-100-33	190.2	174	14.2	3.0	2.5	2.3	1.6	0.2
	Average	**190.2**	**175**	**14.3**	**2.9**	**2.4**	**2.2**	**1.4**	**0.2**
2405	M. R.-135-55	187.8	168	14.3	8.4	6.6	5.9	4.2	0.4
2406	M. R.-135-55	187.8	167	14.2	8.6	6.7	6.0	4.3	0.4
2429	M. R.-135-55	189.4	164	14.3	10.1	7.8	7.0	4.5	0.4
	Average	**188.3**	**166**	**14.3**	**9.0**	**7.0**	**6.3**	**4.3**	**0.4**
2408	Nut-100-33	189.0	178	14.2	3.0	2.4	2.1	1.4	0.2
2409	Nut-100-33	188.5	177	14.3	2.9	2.3	2.0	1.3	0.2
2410	Nut-100-33	186.2	181	14.4	2.9	2.3	2.1	1.4	0.2
2426	Nut-100-33	189.9	182	14.5	3.6	2.8	2.5	1.8	0.1
	Average	**188.4**	**180**	**14.4**	**3.1**	**2.5**	**2.2**	**1.5**	**0.2**
2412	Nut-135-55	187.1	168	14.5	9.2	7.3	6.6	4.6	0.5
2413	Nut-135-55	187.1	168	14.6	9.2	7.2	6.4	4.4	0.5
2414	Nut-135-55	187.5	168	14.5	9.3	7.3	6.5	4.5	0.5
	Average	**187.2**	**168**	**14.5**	**9.2**	**7.3**	**6.5**	**4.5**	**0.5**
2415	Egg-100-33	189.9	180	14.2	3.6	3.1	2.6	1.8	0.2
2416	Egg-100-33	189.5	180	14.2	3.6	3.0	2.6	1.7	0.2
2423	Egg-100-33	190.0	182	14.4	3.3	2.7	2.2	1.4	0.2
	Average	**189.8**	**181**	**14.3**	**3.5**	**2.9**	**2.5**	**1.6**	**0.2**
2420	Egg-135-55	190.1	171	14.2	9.5	7.7	6.7	4.3	0.5
2422	Egg-135-55	189.7	170	14.2	9.2	7.4	6.4	4.0	0.5
2424	Egg-135-55	190.1	170	14.5	9.3	7.3	6.4	4.1	0.5
	Average	**190.0**	**170**	**14.3**	**9.3**	**7.5**	**6.5**	**4.1**	**0.5**
2417	Lump-100-33	189.9	180	14.2	3.6	3.0	2.5	1.7	0.2
2418	Lump-100-33	190.1	180	14.2	3.5	3.0	2.5	1.7	0.2
2419	Lump-100-33	190.0	180	14.4	3.5	3.0	2.5	1.7	0.2
	Average	**190.0**	**180**	**14.3**	**3.5**	**3.0**	**2.5**	**1.7**	**0.2**
2425	Lump-135-55	190.0	168	14.6	9.3	7.5	6.4	4.3	0.5
2427	Lump-135-55	183.4	162	14.4	9.3	7.1	6.3	4.3	0.4
2428	Lump-135-55	186.8	166	14.4	9.4	7.4	6.6	4.4	0.3
2442	Lump-135-55	188.9	166	14.4	10.0	8.0	7.2	4.6	0.5
	Average	**187.3**	**166**	**14.5**	**9.5**	**7.5**	**6.7**	**4.4**	**0.4**
2430	2 in. S.-100-33	188.6	181	14.3	3.8	3.1	3.1	1.9	0.2
2434	2 in. S.-100-33	190.0	181	14.5	3.6	3.0	2.5	....	0.2
2435	2 in. S.-100-33	191.1	182	14.3	3.7	3.1	2.7	2.1	0.2
	Average	**189.9**	**181**	**14.4**	**3.7**	**3.1**	**2.8**	**2.0**	**0.2**
2436	2 in. S.-135-55	185.3	161	14.4	9.4	7.6	6.5	3.7	0.4
2437	2 in. S.-135-55	189.1	162	14.4	9.2	7.4	6.3	3.8	0.3
	Average	**187.2**	**162**	**14.4**	**9.3**	**7.5**	**6.4**	**3.8**	**0.4**
2431	1¼ in. S.-100-33	184.5	178	14.4	3.5	2.9	2.5	1.9	0.2
2432	1¼ in. S.-100-33	189.1	181	14.1	3.6	3.0	2.6	1.1	0.2
2433	1¼ in. S.-100-33	187.7	180	14.4	3.6	3.1	2.6	1.3	0.2
	Average	**187.1**	**180**	**14.3**	**3.6**	**3.0**	**2.6**	**1.4**	**0.2**
2440	1¼ in. S.-135-55	186.6	163	14.4	9.5	7.5	6.6	4.0	0.5
2441	1¼ in. S.-135-55	191.0	166	14.4	9.4	7.5	6.4	3.7	0.5
	Average	**188.8**	**165**	**14.4**	**9.5**	**7.5**	**6.5**	**3.9**	**0.5**

TABLE 14

QUALITY OF STEAM, COAL, CINDERS AND ASH, AND AIR SUPPLY

TEST NUMBER	Laboratory Designation	Quality of Steam in Dome	Degrees of Super-Heat	Factor of Correction for Quality of Steam	Coal Fired Total lb.	Dry Coal Fired Total lb.	Combustible by Analysis Total lb.	Ash by Analysis Total lb.	Stack Cinders Total lb.	Air per lb. of Carbon Consumed lb.	Air per lb. of Coal as Fired lb.
	Code Item ☞	**407**	**409**	**412**	**418**	**419**	**420**	**421**	**423**	......	......
2400	M. R.-100-33	0.9815	194	0.987	11399	10472	9393	1079	360	19.5	12.5
2401	M. R.-100-33	0.9791	190	0.985	20000	18432	16512	1920	592	19.4	12.4
2402	M. R.-100-33	0.9801	187	0.986	17000	15628	13960	1668	529.	18.3	11.7
	Average	**0.9802**	**190**	....	**16133**	....	....	....	**494**	**19.1**	**12.2**
2405	M. R.-135-55	0.9628	254	0.973	20000	18348	16346	2002	1926	15.6	8.9
2406	M. R.-135-55	0.9574	257	0.969	18630	17110	15353	1757	1579	15.7	9.2
2429	M. R.-135-55	0.9437	246	0.960	14000	12824	11390	1434	1247	16.6	9.5
	Average	**0.9546**	**252**	....	**17543**	....	....	....	**1584**	**16.0**	**9.2**
2408	Nut-100-33	0.9890	210	0.992	12955	11853	10702	1150	236	17.0	11.2
2409	Nut-100-33	0.9836	204	0.988	7808	7125	6423	702	169	16.7	10.8
2410	Nut-100-33	0.9840	189	0.989	14731	13442	12100	1342	311	16.6	10.8
2426	Nut-100-33	0.9853	192	0.989	16310	15064	13550	1514	482	16.5	10.8
	Average	**0.9855**	**199**	....	**12951**	....	....	....	**300**	**16.7**	**10.9**
2412	Nut-135-55	0.9516	255	0.965	18683	17022	15371	1652	1103	14.2	8.7
2413	Nut-135-55	0.9470	258	0.962	20811	18938	17196	1742	1191	15.4	9.6
2414	Nut-135-55	0.9460	260	0.961	13884	12647	11452	1195	780	15.8	9.8
	Average	**0.9482**	**258**	....	**17793**	....	....	....	**1025**	**15.1**	**9.4**
2415	Egg-100-33	0.9871	197	0.991	11888	10875	9922	953	273	19.7	12.9
2416	Egg-100-33	0.9885	195	0.992	19915	18087	16372	1715	445	20.5	13.3
2423	Egg-100-33	0.9873	191	0.991	13520	12317	11243	1073	281	18.0	11.8
	Average	**0.9876**	**194**	....	**15108**	....	....	....	**333**	**19.4**	**12.7**
2420	Egg-135-55	0.9466	235	0.962	13882	12666	11514	1152	1001	16.1	9.7
2422	Egg-135-55	0.9485	215	0.963	14996	13684	12564	1120	1014	15.3	9.5
2424	Egg-135-55	0.9466	242	0.962	14000	12767	11645	1121	1067	15.4	9.3
	Average	**0.9472**	**231**	....	**14293**	....	....	....	**1027**	**15.6**	**9.5**
2417	Lump-100-33	0.9882	199	0.992	13753	12470	11243	1227	285	20.0	12.6
2418	Lump-100-33	0.9861	199	0.990	20537	18508	16727	1781	396	19.5	12.3
2419	Lump-100-33	0.9852	199	0.989	13344	12060	10661	1400	312	18.8	11.5
	Average	**0.9865**	**199**	....	**15878**	....	....	....	**331**	**19.4**	**12.1**
2425	Lump-135-55	0.9440	221	0.960	7499	6850	6247	603	545	15.1	9.3
2427	Lump-135-55	0.9490	207	0.963	11775	10700	9617	1083	903	14.0	7.9
2428	Lump-135-55	0.9442	230	0.960	14122	12816	11566	1250	958	14.7	8.7
2442	Lump-135-55	0.9439	243	0.960	15279	13902	12481	1421	1189	15.4	9.0
	Average	**0.9453**	**225**	....	**12169**	....	....	....	**899**	**14.8**	**8.7**
2430	2 in. S.-100-33	0.9803	203	0.986	10000	9054	7991	1063	825	17.4	10.3
2434	2 in. S.-100-33	0.9668	204	0.976	11950	10826	9595	1231	1058	18.3	10.7
2435	2 in. S.-100-33	0.9714	198	0.979	3822	3478	3073	404	360	18.1	10.7
	Average	**0.9728**	**202**	....	**8591**	....	....	....	**748**	**17.9**	**10.6**
2436	2 in. S.-135-55	0.9442	263	0.960	11556	10491	9297	1194	1521	17.1	9.4
2437	2 in. S.-135-55	0.9439	265	0.960	13254	12041	10570	1471	1975	15.0	7.9
	Average	**0.9441**	**264**	....	**12405**	....	....	....	**1748**	**16.1**	**9.3**
2431	1¼in.S.-100-33	0.9794	210	0.985	7635	6998	6281	718	1023	17.1	9.6
2432	1¼in.S.-100-33	0.9801	211	0.986	7813	7129	6276	853	1029	18.7	10.3
2433	1¼in.S.-100-33	0.9681	193	0.977	13218	12087	10750	1336	1430	18.5	10.5
	Average	**0.9759**	**205**	....	**9555**	....	....	....	**1161**	**18.1**	**10.1**
2440	1¼in.S.-135-55	0.9442	232	0.960	14000	12730	11234	1497	2269	14.6	7.5
2441	1¼in.S.-135-55	0.9437	256	0.960	13750	12275	10663	1612	2185	14.8	7.4
	Average	**0.9440**	**244**	....	**13875**	....	....	....	**2227**	**14.7**	**7.5**

TABLE 15

COAL, CINDERS, ASH, SMOKE AND HUMIDITY

Test Number	Laboratory Designation	Stack Cinder Loss Per Cent of Total Coal as Fired	Stack Cinder Loss Per Cent of Total Dry Coal Fired	Ash from Ash Pan				Smoke Per Cent of Blackness by Ringelmann Chart	Humidity Moisture per lb. of Dry Air lb.
				Total lb.	Per Cent of Total Dry Coal Fired	Per Cent of Total Coal as Fired	Per Cent of Ash by Analysis		
	Code Item ☞	....	427	428	429	....	430	431	435
2400	M. R.-100-33	3.2	3.4	335	3.2	2.9	31.1	29	0.016
2401	M. R.-100-33	3.0	3.2	1331	7.2	6.7	69.3	31	0.015
2402	M. R.-100-33	3.1	3.4	587	3.8	3.5	35.2	35	0.020
	Average	**3.1**	**3.3**	....	**4.7**	**4.4**	**45.2**	**32**	....
2405	M. R.-135-55	9.6	10.5	1248	6.8	6.2	62.3	59	0.011
2406	M. R.-135-55	8.5	9.2	1272	7.4	6.8	72.4	57	0.010
2429	M. R.-135-55	8.9	9.7	976	7.6	7.0	68.1	38	0.005
	Average	**9.0**	**9.8**	....	**7.3**	**6.7**	**67.6**	**51**	....
2408	Nut-100-33	1.8	2.0	555	4.7	4.3	48.3	41	0.008
2409	Nut-100-33	2.2	2.4	381	5.4	4.9	54.3	37	0.009
2410	Nut-100-33	2.1	2.3	1029	7.7	7.0	76.7	37	0.009
2426	Nut-100-33	3.0	3.2	789	5.2	4.8	52.1	29	0.007
	Average	**2.3**	**2.5**	....	**5.8**	**5.3**	**57.9**	**36**	....
2412	Nut-135-55	5.9	6.5	1037	6.1	5.6	62.8	58	0.008
2413	Nut-135-55	5.7	6.3	677	3.6	3.3	38.9	50	0.006
2414	Nut-135-55	5.6	6.2	865	6.8	6.2	72.4	49	0.008
	Average	**5.7**	**6.3**	....	**5.5**	**5.0**	**58.0**	**52**	....
2415	Egg-100-33	2.3	2.5	690	6.3	5.8	72.4	1	0.012
2416	Egg-100-33	2.2	2.5	1155	6.4	5.8	67.4	3	0.012
2423	Egg-100-33	2.1	2.3	637	5.2	4.7	59.4	18	0.007
	Average	**2.2**	**2.4**	....	**6.0**	**5.4**	**66.4**	**7**	....
2420	Egg-135-55	7.2	7.9	1041	8.2	7.5	90.4	22	0.009
2422	Egg-135-55	6.8	7.4	982	7.2	6.6	87.7	38	0.010
2424	Egg-135-55	7.6	8.4	1009	7.9	7.2	90.0	39	0.008
	Average	**7.2**	**7.9**	....	**7.8**	**7.1**	**89.4**	**33**	....
2417	Lump-100-33	2.1	2.3	1017	8.2	7.4	82.9	3	0.010
2418	Lump-100-33	1.9	2.1	1520	8.2	7.4	85.4	3	0.010
2419	Lump-100-33	2.3	2.6	1185	9.8	8.9	84.6	7	0.010
	Average	**2.1**	**2.3**	....	**8.7**	**7.9**	**84.3**	**4**	......
2425	Lump-135-55	7.3	8.0	199	2.9	2.7	33.0	45	0.006
2427	Lump-135-55	7.7	8.4	1349	12.6	11.5	124.6	53	0.006
2428	Lump-135-55	6.8	7.5	1008	7.9	7.1	80.6	....	0.005
2442	Lump-135-55	7.8	8.6	1319	9.5	8.6	92.8	48	0.004
	Average	**7.4**	**8.1**	....	**8.2**	**7.7**	**82.8**	**49**	....
2430	2 in. S.-100-33	8.3	9.1	388	4.3	3.9	36.5	22	0.005
2434	2 in. S.-100-33	8.9	9.8	578	5.3	4.8	47.0	23	0.006
2435	2 in. S.-100-33	9.4	10.4	123	3.5	3.2	30.5	28	0.005
	Average	**8.9**	**9.8**	....	**4.4**	**4.0**	**38.0**	**24**	....
2436	2 in. S.-135-55	13.2	14.5	587	5.6	5.1	49.2	52	0.004
2437	2 in. S.-135-55	14.9	16.4	637	5.3	4.8	43.3	58	0.004
	Average	**14.1**	**15.5**	....	**5.5**	**5.0**	**46.3**	**55**	....
2431	1¼ in. S.-100-33	13.4	14.6	378	5.4	5.0	52.7	27	0.010
2432	1¼ in. S.-100-33	13.2	14.4	330	4.6	4.2	38.7	26	0.011
2433	1¼ in. S.-100-33	10.8	11.8	634	5.3	4.8	47.5	30	0.008
	Average	**12.5**	**13.6**	....	**5.1**	**4.7**	**46.3**	**28**	....
2440	1¼ in. S.-135-55	16.2	17.8	740	5.8	5.3	49.4	52	0.004
2441	1¼ in. S.-135-55	15.9	17.8	453	3.7	3.3	28.1	62	0.004
	Average	**16.1**	**17.8**	....	**4.8**	**4.3**	**38.8**	**57**	....

TABLE 16
COAL ANALYSIS

TEST NUMBER	Laboratory Designation	Proximate Analysis—Coal as Fired					Calorific Value per lb. of Coal as Fired B.t.u.	Ultimate Analysis Coal as Fired			
		Fixed Carbon, Per Cent	Volatile Matter, Per Cent	Moisture, Per Cent	Ash, Per Cent	Sulphur Separately Determined, Per Cent		Carbon, Per Cent	Hydrogen, Per Cent	Nitrogen, Per Cent	Oxygen, Per Cent
	Code Item ☛	**437**	**438**	**440**	**441**	**442**	**443**	**449**	**450**	**451**	**452**
2400	M. R.-100-33	48.08	34.32	8.13	9.47	0.93	11929	66.90	4.29	1.56	8.73
2401	M. R.-100-33	48.51	34.05	7.84	9.60	0.95	11992	67.01	4.30	1.56	8.74
2402	M. R.-100-33	46.96	35.16	8.07	9.81	0.98	11885	66.62	4.28	1.55	8.69
	Average	**47.85**	**34.51**	**8.01**	**9.63**	**0.95**	**11935**	**66.84**	**4.29**	**1.56**	**8.72**
2405	M. R.-135-55	47.83	33.90	8.26	10.01	0.81	11752	66.44	4.26	1.55	8.67
2406	M. R.-135-55	48.56	33.85	8.16	9.43	0.92	11876	66.91	4.29	1.56	8.73
2429	M. R.-135-55	47.55	33.81	8.40	10.24	1.09	11806	65.91	4.23	1.53	8.60
	Average	**47.98**	**33.85**	**8.27**	**9.89**	**0.94**	**11811**	**66.42**	**4.26**	**1.55**	**8.67**
2408	Nut-100-33	47.10	35.51	8.51	8.88	0.83	12002	67.60	4.37	1.38	8.43
2409	Nut-100-33	47.14	35.12	8.75	8.99	1.01	11956	67.16	4.34	1.37	8.38
2410	Nut-100-33	46.50	35.91	8.48	9.11	0.85	11918	67.42	4.36	1.38	8.41
2426	Nut-100-33	48.69	34.39	7.64	9.28	0.82	11992	68.00	4.39	1.39	8.48
	Average	**47.36**	**34.23**	**8.35**	**9.07**	**0.88**	**11967**	**67.55**	**4.37**	**1.38**	**8.43**
2412	Nut-135-55	48.28	33.99	8.89	8.84	0.86	11918	67.29	4.35	1.38	8.39
2413	Nut-135-55	48.01	34.62	9.00	8.37	0.92	11990	67.54	4.36	1.38	8.42
2414	Nut-135-55	48.20	34.28	8.91	8.61	0.87	11923	67.46	4.36	1.38	8.41
	Average	**48.16**	**34.30**	**8.93**	**8.61**	**0.88**	**11944**	**67.43**	**4.36**	**1.38**	**8.41**
2415	Egg-100-33	47.89	35.57	8.52	8.02	1.29	12143	68.18	4.49	1.50	7.99
2416	Egg-100-33	48.34	33.87	9.18	8.61	0.93	11918	67.45	4.45	1.49	7.90
2423	Egg-100-33	48.39	34.77	8.90	7.94	0.73	12100	68.40	4.51	1.51	8.01
	Average	**48.21**	**34.74**	**8.87**	**8.19**	**0.98**	**12054**	**68.01**	**4.48**	**1.50**	**7.97**
2420	Egg-135-55	48.65	34.29	8.76	8.30	0.86	12042	68.11	4.49	1.50	7.98
2422	Egg-135-55	49.24	34.54	8.75	7.47	0.92	12175	68.76	4.53	1.52	8.05
2424	Egg-135-55	48.83	34.35	8.81	8.01	0.91	12049	68.27	4.50	1.51	8.00
	Average	**48.91**	**34.39**	**8.77**	**7.93**	**0.90**	**12089**	**68.38**	**4.51**	**1.51**	**8.01**
2417	Lump-100-33	46.84	34.91	9.33	8.92	1.06	11826	66.26	4.22	1.48	8.72
2418	Lump-100-33	46.36	35.09	9.88	8.67	0.83	11794	66.21	4.22	1.48	8.72
2419	Lump-100-33	46.54	33.35	9.62	10.49	1.00	11601	64.78	4.13	1.45	8.53
	Average	**46.58**	**34.45**	**9.61**	**9.36**	**0.96**	**11740**	**65.75**	**4.19**	**1.47**	**8.66**
2425	Lump-135-55	48.06	35.24	8.66	8.04	0.93	12062	67.64	4.31	1.52	8.90
2427	Lump-135-55	47.66	34.01	9.13	9.20	0.89	11776	66.34	4.22	1.49	8.73
2428	Lump-135-55	46.38	35.52	9.25	8.85	0.82	11853	66.58	4.24	1.49	8.76
2442	Lump-135-55	48.58	33.11	9.01	9.30	0.64	11806	66.56	4.24	1.49	8.76
	Average	**47.67**	**34.47**	**9.01**	**8.85**	**0.82**	**11874**	**66.78**	**4.25**	**1.50**	**8.79**
2430	2 in. S.-100-33	47.98	31.93	9.46	10.63	0.84	11542	65.54	4.42	1.47	7.64
2434	2 in. S.-100-33	48.12	32.17	9.41	10.30	0.89	11579	65.81	4.44	1.48	7.67
2435	2 in. S.-100-33	48.33	32.08	9.01	10.58	0.74	11565	66.04	4.45	1.48	7.70
	Average	**48.14**	**32.06**	**9.29**	**10.50**	**0.82**	**11562**	**65.80**	**4.44**	**1.46**	**7.67**
2436	2 in. S.-135-55	48.70	31.75	9.22	10.33	0.79	11592	66.03	4.45	1.48	7.70
2437	2 in. S.-135-55	47.45	32.30	9.15	11.10	0.97	11470	65.30	4.40	1.47	7.61
	Average	**48.05**	**32.03**	**9.19**	**10.72**	**0.88**	**11531**	**65.67**	**4.43**	**1.48**	**7.66**
2431	1¼in.S.-100-33	48.65	33.61	8.34	9.40	0.99	11851	67.06	4.45	1.46	8.30
2432	1¼in.S.-100-33	48.13	32.20	8.75	10.92	1.07	11543	65.40	4.34	1.43	8.09
2433	1¼in.S.-100-33	48.49	32.84	8.56	10.11	0.90	11698	66.36	4.41	1.45	8.21
	Average	**48.42**	**32.88**	**8.55**	**10.14**	**0.99**	**11697**	**66.27**	**4.40**	**1.45**	**8.20**
2440	1¼in.S.-135-55	47.85	32.39	9.07	10.69	0.97	11542	65.41	4.34	1.43	8.09
2441	1¼in.S.-135-55	46.91	30.64	10.73	11.72	0.91	11151	63.24	4.20	1.38	7.82
	Average	**47.38**	**31.52**	**9.90**	**11.21**	**0.94**	**11347**	**64.33**	**4.27**	**1.41**	**7.96**

TABLE 17

CALORIFIC VALUE OF COAL AND CINDERS, ANALYSIS OF FRONT-END GASES

TEST NUMBER	Laboratory Designation	Calorific Value B.t.u. per lb.				Analysis of Front End Gases—per cent					
		Dry Coal	Combustible	Stack Cinders	Ash	Oxygen O_2	Carbon Monoxide CO	Carbon Dioxide CO_2	Nitrogen N_2	Hydrogen H_2	Methane CH_4
	Code Item ☞	**458**	**459**	**462**	**463**	**466**	**467**	**468**	**409**	**470**	**471**
2400	M. R.-100-33	12983	14476	8399	3488	6.47	0.095	12.52	80.95	0.020	0.005
2401	M. R.-100-33	13012	14526	8563	3141	6.32	0.148	12.53	81.02	0.009	0.003
2402	M. R.-100-33	12929	14474	8570	2695	5.55	0.220	13.21	81.02	0.000	0.015
	Average	**12975**	....	**8511**	**3108**	**6.11**	**0.154**	**12.75**	**81.00**	**0.010**	**0.008**
2405	M. R.-135-55	12811	14380	11081	3935	2.79	0.601	15.19	81.26	0.093	0.068
2406	M. R.-135-55	12933	14414	11030	3852	2.15	1.100	14.71	82.00	0.020	0.020
2429	M. R.-135-55	12888	14510	10921	4410	3.70	0.307	14.54	81.38	0.033	0.033
	Average	**12877**	....	**11011**	**4066**	**2.88**	**0.669**	**14.81**	**81.55**	**0.049**	**0.040**
2408	Nut-100-33	13118	14528	8023	2633	4.59	0.440	13.98	80.89	0.070	0.035
2409	Nut-100-33	13102	14535	7585	3204	4.28	0.476	14.22	80.92	0.047	0.053
2410	Nut-100-33	13023	14461	8231	2187	4.11	0.376	14.45	80.97	0.058	0.034
2426	Nut-100-33	12983	14434	8458	1985	4.03	0.420	14.44	80.99	0.087	0.033
	Average	**13057**	....	**8074**	**2502**	**4.25**	**0.428**	**14.27**	**80.94**	**0.066**	**0.039**
2412	Nut-135-55	13081	14485	10728	3416	1.85	1.575	15.65	80.54	0.180	0.205
2413	Nut-135-55	13176	14512	10822	3184	2.41	0.610	15.34	81.00	0.265	0.190
2414	Nut-135-55	13090	14456	10634	3409	2.09	0.645	15.09	81.94	0.155	0.040
	Average	**13116**	....	**10728**	**3336**	**2.12**	**0.943**	**15.36**	**81.16**	**0.200**	**0.145**
2415	Egg-100-33	13273	14549	7987	3173	6.85	0.200	12.21	80.69	0.028	0.028
2416	Egg-100-33	13122	14495	7999	3357	7.12	0.118	11.85	80.92	0.002	0.000
2423	Egg-100-33	13282	14551	8329	3827	5.09	0.184	13.53	81.18	0.008	0.010
	Average	**13226**	....	**8105**	**3452**	**6.35**	**0.167**	**12.53**	**80.93**	**0.013**	**0.013**
2420	Egg-135-55	13198	14519	10771	4651	2.91	0.390	15.01	81.64	0.035	0.025
2422	Egg-135-55	13345	14535	11234	3343	2.43	0.400	15.71	81.30	0.112	0.040
2424	Egg-135-55	13214	14486	10584	4426	2.65	0.450	15.53	81.27	0.060	0.040
	Average	**13252**	....	**10863**	**4140**	**2.66**	**0.413**	**15.42**	**81.40**	**0.069**	**0.035**
2417	Lump-100-33	13043	14467	7713	3469	6.94	0.112	12.16	80.78	0.002	0.008
2418	Lump-100-33	13086	14479	7574	3598	6.49	0.102	12.50	80.92	0.003	0.002
2419	Lump-100-33	12836	14521	7106	3436	5.88	0.170	12.90	80.98	0.015	0.010
	Average	**12988**	....	**7464**	**3501**	**6.44**	**0.128**	**12.52**	**80.89**	**0.007**	**0.007**
2425	Lump-135-55	13205	14479	10917	3487	2.39	0.695	15.61	81.07	0.140	0.110
2427	Lump-135-55	12958	14418	10849	4869	0.90	1.630	15.96	80.94	0.350	0.203
2428	Lump-135-55	13061	14472	10829	4297	1.57	0.865	15.89	81.48	0.140	0.060
2442	Lump-135-55	12974	14450	10415	4527	2.67	0.343	15.66	81.26	0.038	0.033
	Average	**13050**	....	**10753**	**4295**	**1.88**	**0.883**	**15.78**	**81.19**	**0.167**	**0.102**
2430	2 in. S.-100-33	12748	14443	9407	3674	5.01	0.330	13.74	80.81	0.057	0.043
2434	2 in. S.-100-33	12782	14422	9569	3710	5.40	0.157	13.28	81.14	0.003	0.013
2435	2 in. S.-100-33	12710	14382	9113	3677	5.25	0.280	13.34	81.12	0.010	0.000
	Average	**12747**	....	**9363**	**3687**	**5.22**	**0.256**	**13.45**	**81.02**	**0.023**	**0.019**
2436	2 in. S.-135-55	12769	14409	10611	4343	4.25	0.503	13.88	81.20	0.073	0.073
2437	2 in. S.-135-55	12625	14382	11018	4021	2.08	0.650	15.76	81.33	0.137	0.057
	Average	**12697**	....	**10815**	**4182**	**3.17**	**0.577**	**14.82**	**81.27**	**0.105**	**0.065**
2431	1¼in.S.-100-33	12929	14407	10505	4109	4.90	0.340	13.97	80.80	0.000	0.000
2432	1¼in.S.-100-33	12650	14371	10157	4504	5.67	0.227	12.91	81.17	0.017	0.007
2433	1¼in.S.-100-33	12793	14384	10784	4252	5.50	0.228	13.05	81.19	0.050	0.013
	Average	**12791**	....	**10482**	**4288**	**5.36**	**0.265**	**13.31**	**81.05**	**0.022**	**0.007**
2440	1¼in.S.-135-55	12692	14384	10870	3838	2.11	1.380	15.37	80.42	0.313	0.443
2441	1¼in.S.-135-55	12492	14380	11203	4469	1.92	0.760	15.86	81.15	0.177	0.137
	Average	**12592**	....	**11037**	**4154**	**2.02**	**1.070**	**15.62**	**80.79**	**0.245**	**0.290**

TABLE 18

WATER AND DRAWBAR PULL

Test Number	Laboratory Designation	Water						
		Delivered to Boiler by Injectors lb.	Weight of Water in Boiler at Start of Test Minus Weight in Boiler at Close of Test, lb.	Correction for Change of Water Level and Steam Pressure in Boiler, Start to Close, lb.	Loss from Boiler lb.	Loss from Boiler Corrected lb.	Presumably Evaporated lb.	Drawbar Pull lb.
	Code Item ☞	476	477	478	479	480	481	467
2400	M. R.-100-33	80868	+1110	+ 794	0	0	81662	21970
2401	M. R.-100-33	141151	—1110	— 791	0	0	140360	21727
2402	M. R.-100-33	119334	+ 220	+ 157	0	0	119491	21822
	Average	....	....	....	....	....	**113838**	**21840**
2405	M. R.-135-55	113133	+2060	+1377	0	0	114510	28771
2406	M. R.-135-55	112494	+ 160	+ 114	0	0	112608	28718
2429	M. R.-135-55	80329	+2640	+1860	172	121	82068	28672
	Average	....	....	....	....	....	**103062**	**28720**
2408	Nut-100-33	88009	0	— 110	0	0	87899	22490
2409	Nut-100-33	54574	0	— 22	325	232	54320	22411
2410	Nut-100-33	99995	0	— 43	75	54	99898	22417
2426	Nut-100-33	111980	+ 180	+ 128	495	353	111755	22640
	Average	....	....	....	....	....	**88468**	**22490**
2412	Nut-135-55	112833	+2950	+1856	160	113	114576	28958
2413	Nut-135-55	127417	+1380	+ 973	180	127	128263	29100
2414	Nut-135-55	83950	+ 880	+ 573	120	85	84438	29128
	Average	....	....	....	....	....	**109092**	**29062**
2415	Egg-100-33	84089	0	0	315	225	83864	2284
2416	Egg-100-33	139492	0	— 325	525	375	138792	23115
2423	Egg-100-33	96044	+ 180	+ 128	360	257	95915	22533
	Average	....	....	....	....	....	**106190**	**22829**
2420	Egg-135-55	84550	+2740	+1920	120	85	86385	29046
2422	Egg-135-55	91315	+2420	+1707	130	92	92930	29030
2424	Egg-135-55	83485	+3320	+2330	120	85	85730	29104
	Average	....	....	....	....	....	**88348**	**29060**
2417	Lump-100-33	96335	— 590	— 422	360	257	95656	23026
2418	Lump-100-33	138919	+ 340	+ 220	525	375	138764	23085
2419	Lump-100-33	89522	0	0	330	236	89286	22983
	Average	....	....	....	....	....	**107902**	**23031**
2425	Lump-135-55	41743	+1610	+1136	60	42	42837	28530
2427	Lump-135-55	60757	+3860	+2639	135	96	63300	27909
2428	Lump-135-55	77730	+3130	+2125	165	116	79739	28441
2442	Lump-135-55	89495	+ 740	+ 543	90	63	89975	29266
	Average	....	....	....	....	....	**68963**	**28537**
2430	2 in. S.-100-33	63153	+ 350	+ 293	150	107	63339	22906
2434	2 in. S.-100-33	74750	+1070	+ 759	141	100	75409	23091
2435	2 in. S.-100-33	24008	+ 500	+ 376	45	32	24342	23268
	Average	....	....	....	....	....	**54363**	**23088**
2436	2 in. S.-135-55	55754	+1970	+1392	60	42	57104	27976
2437	2 in. S.-135-55	66936	—1040	— 732	68	48	66156	28938
	Average	....	....	....	....	....	**61630**	**28457**
2431	1¼ in. S.-100-33	43907	+ 360	+ 257	105	75	44089	22332
2432	1¼ in. S.-100-33	44696	+2250	+1585	110	79	46202	22912
2433	1¼ in. S -100-33	76873	+ 940	+ 689	180	128	77434	22588
	Average	....	....	....	....	....	**55908**	**22611**
2440	1¼ in. S.-135-55	64217	+ 980	+ 725	68	48	64894	29061
2441	1¼ in. S.-135-55	65872	+ 300	+ 211	68	48	66035	29392
	Average	....	....	....	....	....	**65465**	**29227**

TABLE 19

BOILER PERFORMANCE—COAL AND EVAPORATION

TEST NUMBER	Laboratory Designation	Coal as Fired lb.		Dry Coal Fired—lb.		Evaporation					Super-Heated Steam to Engine per Hour lb.
		Per Hour	Per Hour per Sq. Ft. of Grate Surface	Per Hour	Per Hour per Sq. Ft. of Grate Surface	Moist Steam per Hour lb.	Superheated Steam—lb.				
							Per Hour	Per Hour per Sq. Ft. of Heating Surface	Per lb. of Dry Coal	Per lb. of Coal as Fired	
	Code Item ☞	......	......	**626**	**627**	**633**	......	......	......	......	......
2400	M. R.-100-33	3151	45.1	2895	41.5	22577	22565	4.84	7.79	7.16	22466
2401	M. R.-100-33	3183	45.6	2934	42.0	22340	22328	4.79	7.61	7.01	22257
2402	M. R.-100-33	3269	46.8	3005	43.1	22979	22970	4.93	7.64	7.02	22936
	Average	**3201**	**45.8**	**2945**	**42.2**	**22632**	**22621**	**4.85**	**7.68**	**7.06**	**22533**
2405	M. R.-135-55	7361	105.5	6753	96.8	42145	42176	9.05	6.24	5.73	42365
2406	M. R.-135-55	6944	99.5	6377	91.4	41971	41946	9.00	6.58	6.04	41733
2429	M. R.-135-55	7303	104.6	6690	95.9	42811	42854	9.20	6.41	5.87	43059
	Average	**7203**	**103.2**	**6607**	**94.7**	**42309**	**42325**	**9.08**	**6.41**	**5.88**	**42366**
2408	Nut-100-33	3439	49.3	3146	45.1	23334	23327	5.00	7.42	6.78	23195
2409	Nut-100-33	3347	48.0	3054	43.8	23283	23254	4.99	7.61	6.95	23129
2410	Nut-100-33	3400	48.7	3102	44.4	23055	23059	4.95	7.43	6.78	22962
2426	Nut-100-33	3624	51.9	3348	48.0	24834	24808	5.33	7.41	6.85	24588
	Average	**3453**	**49.5**	**3163**	**45.3**	**23627**	**23612**	**5.07**	**7.47**	**6.84**	**23469**
2412	Nut-135-55	7005	100.4	6382	91.4	42960	42964	9.22	6.73	6.13	42951
2413	Nut-135-55	6937	99.4	6313	90.4	42754	42720	9.17	6.77	6.16	42561
2414	Nut-135-55	6942	99.5	6324	90.6	42219	42209	9.06	6.67	6.08	42224
	Average	**6961**	**99.8**	**6340**	**90.8**	**42644**	**42631**	**9.15**	**6.72**	**6.12**	**42579**
2415	Egg-100-33	3397	48.7	3107	44.5	23961	23944	5.14	7.71	7.05	23665
2416	Egg-100-33	3414	48.9	3101	44.4	23795	23788	5.11	7.67	6.97	23651
2423	Egg-100-33	3380	48.4	3079	44.1	23979	23970	5.14	7.78	7.09	23774
	Average	**3397**	**48.7**	**3096**	**44.3**	**23912**	**23901**	**5.13**	**7.72**	**7.04**	**23697**
2420	Egg-135-55	6941	99.4	6333	90.7	43193	43219	9.28	6.82	6.22	43393
2422	Egg-135-55	6920	99.1	6315	90.5	42884	42881	9.20	6.79	6.20	42953
2424	Egg-135-55	7060	101.2	6438	92.2	43233	43302	9.29	6.73	6.13	43581
	Average	**6974**	**99.9**	**6362**	**91.1**	**43103**	**43134**	**9.26**	**6.78**	**6.18**	**43309**
2417	Lump-100-33	3438	49.3	3118	44.7	23914	23906	5.13	7.67	6.96	23714
2418	Lump-100-33	3521	50.5	3173	45.5	23790	23778	5.10	7.49	6.75	23570
2419	Lump-100-33	3639	52.1	3289	47.1	24348	24340	5.23	7.40	6.69	24190
	Average	**3533**	**50.6**	**3193**	**45.8**	**24017**	**24008**	**5.15**	**7.52**	**6.80**	**23825**
2425	Lump-135-55	7499	107.4	6850	98.1	42837	42889	9.21	6.26	5.72	43160
2427	Lump-135-55	7850	112.5	7133	102.2	42200	42254	9.07	5.92	5.38	42698
2428	Lump-135-55	7704	110.4	6992	100.2	43502	44136	9.47	6.31	5.73	43892
2442	Lump-135-55	7640	109.5	6951	99.6	44988	44910	9.64	6.46	5.88	44441
	Average	**7673**	**110.0**	**6982**	**100.0**	**43382**	**43547**	**9.35**	**6.24**	**5.68**	**43548**
2430	2 in. S.-100-33	3821	54.7	3460	49.6	24203	24122	5.18	6.97	6.31	23516
2434	2 in. S.-100-33	3814	54.6	3455	49.5	24069	24014	5.15	6.95	6.30	23656
2435	2 in. S.-100-33	3952	56.6	3597	51.5	25173	25147	5.40	6.99	6.36	24918
	Average	**3862**	**55.3**	**3504**	**50.2**	**24482**	**24428**	**5.24**	**6.97**	**6.32**	**24030**
2436	2 in. S.-135-55	8560	122.6	7771	111.3	42299	42287	9.07	5.45	4.94	42232
2437	2 in. S.-135-55	8836	126.6	8027	115.0	44104	43981	9.44	5.48	4.98	43376
	Average	**8698**	**124.6**	**7899**	**113.2**	**43202**	**43134**	**9.26**	**5.47**	**4.96**	**42804**
2431	1¼in.S.-100-33	4321	61.9	3960	56.7	24951	24907	5.34	6.29	5.76	24370
2432	1¼in.S.-100-33	4185	60.0	3818	54.7	24747	24714	5.30	6.47	5.90	24469
2433	1¼in.S.-100-33	4264	61.1	3899	55.9	24979	24933	5.35	6.39	5.85	24514
	Average	**4257**	**61.0**	**3892**	**55.8**	**24892**	**24851**	**5.33**	**6.38**	**5.84**	**24451**
2440	1¼in.S.-135-55	9333	133.7	8487	121.6	43263	43186	9.27	5.09	4.63	42746
2441	1¼in.S.-135-55	9167	131.3	8183	117.2	44024	43941	9.43	5.37	4.80	43478
	Average	**9250**	**132.5**	**8335**	**119.5**	**43644**	**43564**	**9.35**	**5.23**	**4.72**	**43112**

TABLE 20

BOILER PERFORMANCE—EVAPORATION AND EQUIVALENT EVAPORATION

TEST NUMBER	Laboratory Designation	Steam Used at Calorimeter, Safety Valve, Leaks, etc.	Super-heated Steam Loss per Hour due to Calorimeter Leaks, Corrections etc. lb.	Dry Coal Loss per Hour Equivalent to Steam Loss lb.	Factor of Evaporation	Equivalent Evaporation from and at 212 Degrees F.—lb.					
						Per Hour	Per Hour, Boiler Excluding Super-Heater	Per Hour, Super-Heater Alone	Per Hour, per Sq. Ft. of Total Heating Surface	Per Hour, per Sq. Ft. of Total Heating Surface Excluding Super-Heater	Per Hour, per Sq. Ft. of Heat-Surface Super-Heater Alone
	Code Item ☛	**638**	....	**643**	**644**	**645**	**646**	**647**	**646**	**649**	**655**
2400	M. R.-100-33	717	+101	+13	1.319	29764	26867	2897	6.39	7.40	2.81
2401	M. R.-100-33	202	+ 78	+10	1.319	29451	26607	2844	6.32	7.33	2.76
2402	M. R.-100-33	288	+ 32	+ 4	1.314	30183	27299	2884	6.48	7.52	2.80
	Average	......	......	......	**1.317**	**29799**	......	......	**6.40**	......	......
2405	M. R.-135-55	87	—191	—31	1.352	57022	49563	7459	12.24	13.65	7.24
2406	M. R.-135-55	685	+199	+30	1.354	56795	49190	7605	12.19	13.55	7.38
2429	M. R.-135-55	426	—212	—33	1.348	57767	49703	8064	12.40	13.69	7.83
	Average	......	......	......	**1.351**	**57195**	......	......	**12.28**	......	......
2408	Nut-100-33	632	+122	+16	1.331	31048	28000	3048	6.66	7.71	2.96
2409	Nut-100-33	613	+128	+17	1.328	30881	27823	3058	6.63	7.66	2.97
2410	Nut-100-33	504	+ 87	+12	1.321	30461	27597	2864	6.54	7.60	2.78
2426	Nut-100-33	1511	+218	+29	1.322	32796	29727	3069	7.04	8.19	2.98
	Average	......	......	......	**1.326**	**31297**	......	......	**6.72**	......	......
2412	Nut-135-55	871	+ 4	+ 1	1.353	58130	50135	7995	12.47	13.81	7.76
2413	Nut-135-55	1115	+154	+23	1.356	57929	49766	8163	12.43	13.71	7.93
2414	Nut-135-55	383	— 8	— 1	1.356	57236	49059	8177	12.28	13.51	7.94
	Average	......	......	......	**1.355**	**57765**	......	......	**12.39**	......	......
2415	Egg-100-33	1259	+270	+35	1.323	31678	28705	2973	6.80	7.91	2.89
2416	Egg-100-33	1537	+136	+18	1.322	31448	28530	2918	6.75	7.86	2.83
2423	Egg-100-33	1130	+184	+24	1.321	31665	28751	2914	6.79	7.92	2.83
	Average	......	......	......	**1.322**	**31597**	......	......	**6.78**	......	......
2420	Egg-135-55	505	—165	—24	1.343	58043	50233	7810	12.46	13.84	7.58
2422	Egg-135-55	657	— 75	—11	1.333	57160	49960	7200	12.27	13.76	6.99
2424	Egg-135-55	384	—292	—43	1.347	58328	50323	8005	12.52	13.86	7.77
	Average	......	......	......	**1.341**	**57644**	......	......	**12.42**	......	......
2417	Lump-100-33	890	+183	+24	1.324	31652	28673	2979	6.79	7.90	2.89
2418	Lump-100-33	1778	+200	+27	1.323	31458	28453	3005	6.75	7.84	2.92
2419	Lump-100-33	816	+145	+20	1.324	32227	29121	3106	6.92	8.02	3.02
	Average	......	......	......	**1.324**	**31779**	......	......	**6.82**	......	......
2425	Lump-135-55	193	—284	—45	1.334	57214	49691	7523	12.28	13.69	7.30
2427	Lump-135-55	570	—436	—74	1.327	56071	49164	6907	12.03	13.54	6.71
2428	Lump-135-55	406	—334	—53	1.341	58403	50549	7854	12.53	13.93	7.63
2442	Lump-135-55	1354	+475	+74	1.348	60539	52276	8263	12.99	14.40	8.02
	Average	......	......	......	**1.338**	**58057**	......	......	**12.46**	......	......
2430	2 in. S.-100-33	1961	+611	+88	1.327	32010	28826	3184	6.87	7.94	3.09
2434	2 in. S.-100-33	1704	+360	+52	1.329	31914	28426	3488	6.85	7.83	3.39
2435	2 in. S.-100-33	412	+226	+32	1.323	33270	29780	3490	7.14	8.20	3.39
	Average	......	......	......	**1.326**	**32398**	......	......	**6.95**	......	......
2436	2 in. S.-135-55	711	+ 62	+11	1.359	57468	49194	8274	12.33	13.55	8.03
2437	2 in. S.-135-55	832	+614	+112	1.360	59814	51249	8565	12.84	14.12	8.32
	Average	......	......	......	**1.360**	**58641**	......	......	**12.59**	......	......
2431	1¼in.S.-100-33	1206	+526	+84	1.329	33102	29692	3410	7.10	8.18	3.31
2432	1¼in.S.-100-33	1262	+249	+38	1.330	32870	29448	3422	7.05	8.11	3.32
2433	1¼in.S.-100-33	1818	+418	+65	1.322	32961	29525	3436	7.07	8.13	3.34
	Average	......	......	......	**1.327**	**32978**	......	......	**7.07**	......	......
2440	1¼in.S.-135-55	1079	+439	+86	1.342	57956	50272	7684	12.44	13.85	7.46
2441	1¼in.S.-135-55	955	+457	+85	1.355	59540	51156	8384	12.78	14.09	8.14
	Average	......	......	......	**1.349**	**58748**	......	......	**12.61**	......	......

TABLE 21

BOILER PERFORMANCE —HEAT TRANSFER, EQUIVALENT EVAPORATION, HORSE POWER AND EFFICIENCY

Test Number	Laboratory Designation	Heat Transfer across Water H. S. per Min. B.t.u.	Heat Transfer across Super-Heater H. S. per Min. B.t.u.	Per Cent of Evaporation by		Equivalent Evaporation from and at 212 degrees F.—lb.				Boiler Horse-Power	Efficiency of Boiler Per Cent
				Water Heating Surface	Super-Heating Surface	Per Hour per Sq. Ft. of Grate Area	Per lb. of Coal as Fired	Per lb. of Dry Coal	Per lb. of Combustible		
	Code Item ☞	....	....	....	....	656	657	658	659	660	666
2400	M. R.-100-33	435096	46917	90.3	9.7	426.4	9.44	10.28	11.46	863	76.89
2401	M. R.-100-33	430864	46072	90.3	9.7	421.9	9.25	10.04	11.21	854	74.95
2402	M. R.-100-33	442231	46713	90.4	9.6	432.4	9.23	10.04	11.24	875	75.46
	Average	**436064**	**46567**	....	....	....	**9.31**	**10.12**	....	**864**	**75.77**
2405	M. R.-135-55	802652	120811	86.9	13.1	817.0	7.75	8.44	9.48	1653	64.08
2406	M. R.-135-55	796819	123182	86.6	13.4	813.7	8.18	8.91	9.93	1646	66.93
2429	M. R.-135-55	804918	130612	86.0	14.0	827.6	7.91	8.64	9.72	1674	65.10
	Average	**801463**	**124868**	....	....	....	**7.95**	**8.66**	....	**1658**	**65.37**
2408	Nut-100-33	453535	49367	90.2	9.8	444.8	9.03	9.87	10.93	900	73.11
2409	Nut-100-33	450798	49535	90.1	9.9	442.4	9.23	10.11	11.22	895	75.02
2410	Nut-100-33	446844	46383	90.6	9.4	436.4	8.96	9.82	10.91	883	73.05
2426	Nut-100-33	481531	49709	90.6	9.4	469.9	9.05	9.80	10.89	951	73.33
	Average	**458177**	**48749**	....	....	....	**9.07**	**9.90**	....	**907**	**73.63**
2412	Nut-135-55	812016	129497	86.2	13.8	832.8	8.30	9.11	10.09	1685	67.67
2413	Nut-135-55	806198	132223	85.9	14.1	829.9	8.35	9.18	10.11	1679	67.67
2414	Nut-135-55	794562	132443	85.7	14.3	820.0	8.24	9.05	10.00	1659	67.15
	Average	**804259**	**131388**	....	....	....	**8.30**	**9.11**	....	**1674**	**67.50**
2415	Egg-100-33	464843	48158	90.6	9.4	453.9	9.33	10.20	11.17	918	74.66
2416	Egg-100-33	461940	47263	90.7	9.3	450.6	9.21	10.14	11.20	912	75.09
2423	Egg-100-33	465632	47191	90.8	9.2	453.7	9.37	10.28	11.27	918	75.25
	Average	**464138**	**47537**	....	....	....	**9.30**	**10.21**	....	**916**	**75.00**
2420	Egg-135-55	813324	126491	86.5	13.5	831.6	8.36	9.16	10.08	1682	67.46
2422	Egg-135-55	809436	116617	87.4	12.6	818.9	8.26	9.05	9.86	1657	65.92
2424	Egg-135-55	814798	129653	86.3	13.7	835.7	8.26	9.06	9.93	1691	66.61
	Average	**812519**	**124254**	....	....	....	**8.29**	**9.09**	....	**1677**	**66.66**
2417	Lump-100-33	464290	48258	90.6	9.4	453.5	9.21	10.15	11.26	917	75.68
2418	Lump-100-33	460773	48672	90.4	9.6	450.7	8.93	9.91	10.97	912	73.57
2419	Lump-100-33	471540	50315	90.4	9.6	461.7	8.86	9.80	11.08	934	74.21
	Average	**465534**	**49082**	....	....	....	**9.00**	**9.95**	....	**921**	**74.49**
2425	Lump-135-55	804835	121855	86.9	13.1	819.7	7.63	8.35	9.16	1658	61.47
2427	Lump-135-55	796173	111869	87.7	12.3	803.3	7.14	7.86	8.75	1625	58.92
2428	Lump-135-55	818563	127214	86.6	13.4	836.7	7.58	8.35	9.26	1693	62.14
2442	Lump-135-55	846524	133841	86.4	13.6	867.3	7.92	8.71	9.70	1755	65.19
	Average	**816524**	**123695**	....	....	....	**7.57**	**8.32**	......	**1684**	**61.93**
2430	2 in. S.-100-33	466957	51578	90.1	9.9	458.6	8.38	9.25	10.48	928	70.55
2434	2 in. S.-100-33	460520	56498	89.1	10.9	457.2	8.37	9.24	10.42	925	70.24
2435	2 in. S.-100-33	482315	56522	89.5	10.5	476.7	8.42	9.25	10.47	964	70.75
	Average	**469931**	**54866**	....	....	....	**8.39**	**9.25**	......	**939**	**70.51**
2436	2 in. S.-135-55	796631	134016	85.6	14.4	823.3	6.71	7.40	8.34	1666	56.25
2437	2 in. S.-135-55	830111	138731	85.7	14.3	857.0	6.77	7.45	8.49	1734	57.35
	Average	**813371**	**136374**	....	....	....	**6.74**	**7.43**	....	**1700**	**56.80**
2431	1¼in.S.-100-33	480723	55239	89.7	10.3	474.3	7.66	8.36	9.31	959	62.81
2432	1¼in.S.-100-33	476916	55422	89.6	10.4	470.9	7.85	8.61	9.78	953	66.08
2433	1¼in.S.-100-33	478098	55647	89.6	10.4	472.2	7.73	8.45	9.51	955	64.21
	Average	**478579**	**55436**	....	....	....	**7.75**	**8.47**	....	**956**	**64.37**
2440	1¼in.S.-135-55	814282	124462	86.7	13.3	830.3	6.21	6.83	7.74	1680	52.28
2441	1¼in.S.-135-55	828679	135796	85.9	14.1	853.0	6.50	7.28	8.38	1726	56.64
	Average	**821481**	**130129**	....	....	....	**6.36**	**7.06**	....	**1703**	**54.46**

TABLE 22

ENGINE PERFORMANCE

Test Number	Laboratory Designation	Cut-Off Per Cent of Stroke Average	Least Back-Pressure lb. per Sq. in. Average	Mean Effective Pressure lb. per Sq. in. Average	Indicated Horse Power				
					Right Side		Left Side		Total
					Head End	Crank End	Head End	Crank End	
	Code Item ☞	....	....	678	707	708	709	710	711
2400	M. R.-100-33	....	....	....	....	....	....	....	....
2401	M. R.-100-33	34.0	1.9	74.1	308.0	321.3	296.9	298.3	1224.5
2402	M. R.-100-33	33.0	2.1	73.7	309.3	320.7	296.1	297.5	1223.6
	Average	**33.5**	**2.0**	**73.9**	....	....	....	....	**1224.1**
2405	M. R.-135-55	54.3	11.7	96.9	531.8	567.2	529.2	529.5	2157.7
2406	M. R.-135-55	53.7	12.0	95.9	528.7	562.8	527.0	533.0	2151.5
2429	M. R.-135-55	55.9	12.9	97.5	529.9	572.3	546.2	542.6	2191.0
	Average	**54.6**	**12.2**	**96.8**	....	....	....	....	**2166.7**
2408	Nut-100-33	33.0	2.1	77.9	327.9	342.0	309.0	314.7	1293.6
2409	Nut-100-33	32.4	2.6	77.8	327.6	338.2	310.6	310.1	1286.5
2410	Nut-100-33	31.5	2.3	77.2	326.6	338.3	304.9	311.1	1280.9
2426	Nut-100-33	31.7	2.7	79.6	332.7	347.1	312.7	321.2	1313.7
	Average	**32.2**	**2.4**	**78.1**	....	....	....	....	**1293.7**
2412	Nut-135-55	....	....	....	....	....	....	....	....
2413	Nut-135-55	55.7	11.7	98.0	547.6	576.3	536.8	539.6	2200.3
2414	Nut-135-55	59.3	11.9	98.9	551.5	581.5	542.0	546.8	2221.8
	Average	**57.5**	**11.8**	**98.5**	....	....	....	....	**2211.1**
2415	Egg-100-33	32.4	2.7	79.6	331.7	350.0	311.7	320.3	1313.7
2416	Egg-100-33	33.3	2.4	80.0	334.2	351.5	317.4	321.2	1324.3
2423	Egg-100-33	32.2	2.4	78.4	330.1	336.5	312.6	311.9	1291.1
	Average	**32.6**	**2.5**	**79.3**	....	....	....	....	**1309.7**
2420	Egg-135-55	57.2	13.0	97.3	540.4	571.5	536.3	540.5	2188.7
2422	Egg-135-55	58.2	13.0	97.9	552.5	580.2	533.2	548.4	2214.3
2424	Egg-135-55	56.6	12.9	98.3	547.6	585.8	544.0	542.6	2220.0
	Average	**57.3**	**13.0**	**97.8**	....	....	....	....	**2207.7**
2417	Lump-100-33	36.3	2.2	80.0	335.5	350.3	320.1	317.3	1323.2
2418	Lump-100-33	33.5	2.6	80.1	338.8	350.8	320.0	320.0	1329.6
2419	Lump-100-33	32.7	2.8	79.7	335.0	351.9	314.4	319.7	1321.0
	Average	**34.2**	**2.5**	**79.9**	....	....	....	....	**1324.6**
2425	Lump-135-55	56.0	12.2	97.8	546.5	572.3	535.5	547.1	2201.4
2427	Lump-135-55	55.7	12.5	96.2	533.3	566.8	532.1	537.5	2169.7
2428	Lump-135-55	55.1	12.3	97.8	541.3	571.4	543.1	544.7	2200.5
2442	Lump-135-55	56.5	12.0	98.5	546.1	576.5	535.4	537.5	2198.5
	Average	**55.8**	**12.3**	**97.6**	....	....	....	....	**2192.5**
2430	2 in. S.-100-33	32.6	2.5	78.5	328.8	345.5	313.6	317.0	1304.9
2434	2 in. S.-100-33	33.6	2.8	80.3	337.3	345.0	321.0	311.2	1314.5
2435	2 in. S.-100-33	34.9	3.0	82.1	352.9	360.0	331.9	320.4	1365.2
	Average	**33.7**	**2.8**	**80.3**	....	....	....	....	**1328.2**
2436	2 in. S.-135-55	56.8	12.4	95.2	527.2	559.3	524.4	515.4	2126.3
2437	2 in. S.-135-55	56.9	12.7	100.0	563.6	585.5	550.9	543.5	2243.5
	Average	**56.9**	**12.6**	**97.6**	....	....	....	....	**2184.9**
2431	1¼ in. S.-100-33	33.9	2.9	79.9	337.2	353.8	321.9	317.0	1329.9
2432	1¼ in. S.-100-33	34.0	3.3	80.8	339.4	356.3	323.3	324.1	1343.1
2433	1¼ in. S.-100-33	33.2	3.0	79.1	334.3	342.6	318.7	314.6	1310.2
	Average	**33.7**	**3.1**	**79.9**	....	....	....	....	**1327.7**
2440	1¼ in. S.-135-55	58.2	11.4	99.3	544.5	580.1	546.7	544.3	2215.6
2441	1¼ in. S.-135-55	55.6	12.2	99.7	550.5	587.3	544.8	549.0	2231.6
	Average	**56.9**	**11.8**	**99.5**	....	....	....	....	**2223.6**

TABLE 23

GENERAL LOCOMOTIVE PERFORMANCE

TEST NUMBER	Laboratory Designation	Consumed per I. H. P. per Hour		Draw-bar Horse-Power	Consumed per D. H. P. per Hour		Tractive Force Based on M.E.P. lb.	Machine Friction in Terms of Horse-Power	Machine Efficiency of Locomotive Per Cent	Efficiency of Locomotive Per Cent
		Dry Coal lb.	Super-Heated Steam lb.		Dry Coal lb.	Super-Heated Steam lb.				
	Code Item ☞	**734**	**740**	**743**	**744**	**747**	**764**	**770**	**778**	**779**
2400	M. R.-100-33	......	......	1108.6	2.60	20.27	......	......	......	7.55
2401	M. R.-100-33	2.39	18.18	1095.2	2.67	20.32	24288	129.3	89.4	7.33
2402	M. R.-100-33	2.46	18.74	1104.6	2.72	20.76	24166	119.0	90.3	7.25
	Average	**2.43**	**18.46**	**1102.8**	**2.66**	**20.45**	**24277**	**124.2**	**89.9**	**7.38**
2405	M. R.-135-55	3.14	19.63	1954.4	3.48	21.68	31766	203.3	90.6	5.72
2406	M. R.-135-55	2.95	19.40	1963.8	3.23	21.25	31457	187.7	91.3	6.09
2429	M. R.-135-55	3.07	19.65	1965.2	3.42	21.91	31962	225.8	89.7	5.78
	Average	**3.05**	**19.56**	**1961.1**	**3.38**	**21.61**	**31728**	**205.6**	**90.5**	**5.86**
2408	Nut-100-33	2.42	17.93	1138.4	2.75	20.38	25563	155.2	88.0	7.07
2409	Nut-100-33	2.36	17.98	1129.7	2.69	20.47	25524	156.8	87.8	7.24
2410	Nut-100-33	2.41	17.93	1133.5	2.73	20.26	25333	147.4	88.5	7.16
2426	Nut-100-33	2.53	18.72	1139.9	2.91	21.57	26088	173.8	86.8	6.73
	Average	**2.43**	**18.14**	**1135.4**	**2.77**	**20.67**	**25627**	**158.3**	**87.8**	**7.05**
2412	Nut-135-55	......	......	1988.7	3.21	21.60	......	......	......	6.07
2413	Nut-135-55	2.86	19.34	1993.0	3.16	21.36	32134	207.3	90.6	6.12
2414	Nut-135-55	2.85	19.00	1994.9	3.17	21.17	32438	226.9	89.8	6.14
	Average	**2.86**	**19.17**	**1992.2**	**3.18**	**21.38**	**32286**	**217.1**	**90.2**	**6.11**
2415	Egg-100-33	2.34	18.01	1150.1	2.67	20.58	26091	163.6	87.6	7.19
2416	Egg-100-33	2.33	17.86	1167.6	2.64	20.26	26208	156.7	88.2	7.34
2423	Egg-100-33	2.36	18.41	1131.0	2.70	21.02	25725	160.1	87.6	7.10
	Average	**2.34**	**18.09**	**1149.6**	**2.67**	**20.62**	**26008**	**160.1**	**87.8**	**7.21**
2420	Egg-135-55	2.90	19.83	1991.6	3.19	21.79	31922	197.1	91.0	6.04
2422	Egg-135-55	2.86	19.40	2001.4	3.17	21.46	32123	212.9	90.4	6.03
2424	Egg-135-55	2.92	19.63	2003.4	3.23	21.75	32256	216.6	90.2	5.96
	Average	**2.89**	**19.62**	**1998.8**	**3.20**	**21.67**	**32100**	**208.9**	**90.5**	**6.01**
2417	Lump-100-33	2.34	17.92	1160.7	2.67	20.43	26257	162.5	87.7	7.31
2418	Lump-100-33	2.37	17.73	1168.6	2.70	20.17	26257	161.0	87.9	7.22
2419	Lump-100-33	2.48	18.31	1162.2	2.81	20.81	26116	155.8	88.0	7.05
	Average	**2.40**	**17.99**	**1163.8**	**2.73**	**20.47**	**26210**	**159.8**	**87.9**	**7.19**
2425	Lump-135-55	3.13	19.61	1957.8	3.52	22.05	32076	243.6	88.9	5.47
2427	Lump-135-55	3.32	19.68	1919.6	3.76	22.24	31554	250.1	88.5	5.23
2428	Lump-135-55	3.20	19.95	1963.8	3.59	22.35	31869	236.7	89.2	5.44
2442	Lump-135-55	3.14	20.24	1989.6	3.45	22.34	32339	208.9	90.5	5.68
	Average	**3.20**	**19.87**	**1957.7**	**3.58**	**22.25**	**31960**	**234.8**	**89.3**	**5.48**
2430	2 in. S.-100-33	2.58	18.02	1160.7	2.90	20.26	25743	144.2	89.0	6.88
2434	2 in. S.-100-33	2.59	18.00	1152.2	2.96	20.53	26342	162.3	87.7	6.74
2435	2 in. S.-100-33	2.61	18.25	1179.1	3.02	21.13	26932	186.1	86.4	6.63
	Average	**2.59**	**18.09**	**1164.0**	**2.36**	**20.64**	**26339**	**164.2**	**87.7**	**6.75**
2436	2 in. S.-135-55	3.64	19.86	1905.6	4.07	22.16	31207	220.7	89.6	4.90
2437	2 in. S.-135-55	3.53	19.33	1980.4	3.99	21.90	32780	263.1	88.3	5.05
	Average	**3.59**	**19.60**	**1943.0**	**4.03**	**22.03**	**31994**	**241.9**	**89.0**	**4.98**
2431	1¼ in. S.-100-33	2.92	18.32	1133.4	3.42	21.50	26213	196.5	85.2	5.77
2432	1¼ in. S.-100-33	2.81	18.22	1161.6	3.26	21.06	26498	181.5	86.5	6.18
2433	1¼ in. S.-100-33	2.93	18.71	1141.0	3.36	21.48	25937	169.2	87.1	5.92
	Average	**2.89**	**18.42**	**1145.3**	**3.35**	**21.35**	**26216**	**182.4**	**86.3**	**5.96**
2440	1¼ in. S.-135-55	3.79	19.29	1977.2	4.25	21.62	32568	238.4	89.2	4.73
2441	1¼ in. S.-135-55	3.63	19.48	2006.7	4.04	21.67	32682	224.9	89.9	5.05
	Average	**3.71**	**19.39**	**1992.0**	**4.15**	**21.65**	**32625**	**231.7**	**89.6**	**4.89**

TABLE 24

ANALYSIS OF ASH AND STACK CINDERS

Test Number	Laboratory Designation	Analysis of Ash				Analysis of Stack Cinders			
		Fixed Carbon, Per Cent	Volatile Matter, Per Cent	Ash, Per Cent	Moisture, Per Cent	Fixed Carbon, Per Cent	Volatile Matter, Per Cent	Ash, Per Cent	Moisture, Per Cent
	Code Item ☞	....	....	**832**	**833**	....	....	**847**	**848**
2400	M. R.-100-33	19.92	4.63	74.11	1.34	53.88	4.87	40.27	0.98
2401	M. R.-100-33	18.30	2.75	78.85	0.10	55.49	4.28	39.48	0.75
2402	M. R.-100-33	15.42	3.16	81.07	0.35	55.47	4.05	39.78	0.70
	Average	**17.88**	**3.51**	**78.01**	**0.60**	**54.95**	**4.40**	**39.64**	**0.81**
2405	M. R.-135-55	24.48	3.34	71.92	0.26	73.75	4.08	21.98	0.19
2406	M. R.-135-55	25.34	2.11	71.77	0.78	73.86	2.91	22.80	0.43
2429	M. R.-135-55	27.83	3.35	67.97	0.85	72.91	3.37	23.47	0.25
	Average	**25.88**	**2.93**	**70.55**	**0.63**	**73.51**	**3.45**	**22.75**	**0.29**
2408	Nut-100-33	16.64	2.26	76.82	4.28	51.75	4.73	42.27	1.25
2409	Nut-100-33	21.25	2.50	73.68	2.57	48.87	4.47	42.69	3.97
2410	Nut-100-33	14.03	2.08	82.56	1.33	52.04	5.57	40.94	1.45
2426	Nut-100-33	13.65	1.70	77.29	7.36	56.35	3.55	39.08	1.02
	Average	**16.39**	**2.14**	**77.59**	**3.89**	**52.25**	**4.58**	**41.25**	**1.92**
2412	Nut-135-55	21.86	1.50	76.39	0.25	74.99	1.87	23.02	0.12
2413	Nut-135-55	21.35	1.83	75.32	1.50	73.36	2.15	24.37	0.12
2414	Nut-135-55	22.43	1.45	75.92	0.20	71.77	2.80	25.39	0.04
	Average	**21.88**	**1.59**	**75.88**	**0.65**	**73.37**	**2.27**	**24.26**	**0.09**
2415	Egg-100-33	19.88	2.51	77.26	0.35	51.03	4.10	44.84	0.03
2416	Egg-100-33	21.28	2.62	73.93	2.17	52.30	4.33	43.00	0.37
2423	Egg-100-33	24.21	2.70	72.38	0.71	55.04	3.94	39.13	1.89
	Average	**21.79**	**2.61**	**74.52**	**1.08**	**52.79**	**4.12**	**42.32**	**0.76**
2420	Egg-135-55	31.55	2.00	66.23	0.22	74.44	2.67	22.78	0.11
2422	Egg-135-55	22.16	1.32	76.26	0.26	77.32	2.04	20.60	0.04
2424	Egg-135-55	28.48	2.27	69.20	0.05	71.82	3.31	24.70	0.17
	Average	**27.40**	**1.86**	**70.56**	**0.18**	**74.53**	**2.67**	**22.69**	**0.11**
2417	Lump-100-33	22.39	1.95	75.55	0.11	48.94	5.02	45.49	0.55
2418	Lump-100-33	21.64	4.14	72.54	1.68	48.27	4.88	46.26	0.59
2419	Lump-100-33	21.31	2.88	74.06	1.75	44.59	5.11	48.59	1.71
	Average	**21.78**	**2.99**	**74.05**	**1.18**	**47.27**	**5.00**	**46.78**	**0.95**
2425	Lump-135-55	22.45	2.39	67.76	7.40	73.00	2.84	24.05	0.11
2427	Lump-135-55	32.52	1.42	65.98	0.08	73.56	2.88	23.28	0.28
2428	Lump-135-55	28.99	1.38	69.58	0.05	73.61	2.26	23.93	0.20
2442	Lump-135-55	29.61	2.38	66.77	1.24	69.58	3.22	26.92	0.28
	Average	**28.39**	**1.89**	**67.52**	**2.19**	**72.44**	**2.80**	**24.55**	**0.22**
2430	2 in. S.-100-33	20.33	5.64	65.86	8.17	61.08	5.43	32.37	1.12
2434	2 in. S.-100-33	19.71	6.52	71.22	2.55	61.57	5.85	31.75	0.83
2435	2 in. S.-100-33	19.30	6.69	57.81	16.20	58.19	6.24	34.14	1.43
	Average	**19.78**	**6.28**	**64.96**	**8.97**	**60.28**	**5.84**	**32.75**	**1.13**
2436	2 in. S.-135-55	26.39	4.31	68.14	1.16	66.06	8.14	25.64	0.16
2437	2 in. S.-135-55	23.16	5.26	70.09	1.49	63.32	13.67	19.88	3.13
	Average	**24.78**	**4.79**	**69.12**	**1.33**	**64.69**	**10.91**	**22.76**	**1.65**
2431	1¼ in. S.-100-33	21.38	7.66	69.41	1.55	65.98	7.13	26.44	0.45
2432	1¼ in. S.-100-33	21.66	10.17	61.67	6.50	63.95	7.06	28.48	0.51
2433	1¼ in. S.-100-33	23.01	7.04	67.85	2.10	69.86	5.90	23.75	0.49
	Average	**22.02**	**8.29**	**66.31**	**3.38**	**66.60**	**6.70**	**26.22**	**0.48**
2440	1¼ in. S.-135-55	21.23	5.89	69.56	3.32	67.35	8.54	23.87	0.24
2441	1¼ in. S.-135-55	24.50	7.09	66.24	2.17	69.57	8.47	21.50	0.46
	Average	**22.87**	**6.49**	**67.90**	**2.75**	**68.46**	**8.51**	**22.69**	**0.35**

TABLE 25

HEAT BALANCE—BRITISH THERMAL UNITS

TEST NUMBER	Laboratory Designation	B.t.u. Absorbed by Boiler per lb. of Coal as Fired	B.t.u. LOSS PER POUND OF COAL AS FIRED									
			Due to Moisture in Coal	Due to Moisture in Air	Due to Hydrogen in Coal	Due to Escaping Gases	Due to Incomplete Combustion			Due to Combustible in Stack Cinders	Due to Combustible in Ash	Due to Radiation, and Unaccounted for
							CO	H_2	CH			
	Code Item ☞	**851**	**852**	**853**	**854**	**855**	......	......	......	**858**	**860**	**869**
2400	M. R.-100-33	9173	102	42	483	1419	49	11	8	265	103	275
2401	M. R.-100-33	8988	99	41	487	1408	76	5	5	254	209	421
2402	M. R.-100-33	8969	102	52	482	1331	107	0	23	267	93	460
	Average	**9043**	**101**	**45**	**484**	**1386**	**77**	**5**	**12**	**262**	**135**	**385**
2405	M. R.-135-55	7531	109	25	487	1239	222	35	78	1067	246	714
2406	M. R.-135-55	7949	108	24	506	1286	416	8	24	934	263	358
2429	M. R.-135-55	7686	112	13	498	1342	120	14	40	973	307	701
	Average	**7722**	**110**	**21**	**497**	**1289**	**253**	**19**	**47**	**991**	**272**	**591**
2408	Nut-100-33	8775	112	23	503	1505	204	33	51	146	113	539
2409	Nut-100-33	8969	114	24	498	1430	214	20	74	164	156	292
2410	Nut-100-33	8706	110	24	497	1367	168	27	47	174	153	646
2426	Nut-100-33	8794	99	18	499	1355	189	39	46	250	96	608
	Average	**8811**	**109**	**42**	**499**	**1414**	**194**	**30**	**55**	**184**	**130**	**521**
2412	Nut-135-55	8065	117	17	475	1199	568	66	230	633	190	358
2413	Nut-135-55	8114	119	16	471	1336	241	106	234	619	104	630
2414	Nut-135-55	8007	118	21	504	1384	258	63	50	598	212	707
	Average	**8062**	**118**	**18**	**483**	**1306**	**356**	**78**	**171**	**617**	**169**	**565**
2415	Egg-100-33	9066	109	36	506	1526	108	16	47	184	184	362
2416	Egg-100-33	8949	117	35	512	1564	66	0	0	178	195	301
2423	Egg-100-33	9105	115	18	521	1437	90	5	15	173	180	440
	Average	**9040**	**114**	**30**	**513**	**1509**	**88**	**7**	**21**	**178**	**186**	**368**
2420	Egg-135-55	8123	114	22	520	1259	178	14	31	777	349	656
2422	Egg-135-55	8026	116	25	526	1355	157	44	49	759	219	899
2424	Egg-135-55	8026	117	20	527	1324	173	23	48	807	319	665
	Average	**8058**	**116**	**22**	**524**	**1313**	**169**	**27**	**43**	**781**	**296**	**740**
2417	Lump-100-33	8949	120	29	486	1519	59	0	13	160	256	236
2418	Lump-100-33	8677	127	29	482	1480	52	3	3	146	266	529
2419	Lump-100-33	8609	128	27	493	1411	82	7	15	166	305	358
	Average	**8745**	**125**	**28**	**487**	**1470**	**64**	**3**	**10**	**157**	**276**	**374**
2425	Lump-135-55	7414	115	16	489	1309	266	54	131	794	92	1382
2427	Lump-135-55	6938	121	12	499	1115	530	12	21	831	558	1139
2428	Lump-135-55	7366	124	13	493	1274	311	51	67	734	307	1113
2442	Lump-135-55	7696	120	10	502	1305	127	15	38	810	391	792
	Average	**7354**	**120**	**13**	**496**	**1251**	**309**	**33**	**64**	**792**	**337**	**1107**
2430	2 in. S.-100-33	8143	122	13	500	1252	141	24	57	776	143	372
2434	2 in. S.-100-33	8133	121	15	513	1303	70	2	18	847	180	378
2435	2 in. S.-100-33	8182	115	11	512	1283	124	4	0	858	118	357
	Average	**8153**	**119**	**13**	**508**	**1279**	**112**	**10**	**25**	**827**	**147**	**369**
2436	2 in. S.-135-55	6520	124	11	520	1357	195	29	88	1396	221	1131
2437	2 in. S.-135-55	6578	122	9	514	1150	212	45	58	1642	193	946
	Average	**6549**	**123**	**10**	**517**	**1254**	**204**	**37**	**73**	**1519**	**207**	**1039**
2431	1¼in.S.-100-33	7443	107	21	513	1160	136	0	0	1408	203	860
2432	1¼in.S.-100-33	7628	111	25	492	1180	97	6	9	1337	190	468
2433	1¼in.S.-100-33	7511	110	19	502	1252	100	22	18	1167	204	793
	Average	**7527**	**109**	**22**	**502**	**1197**	**111**	**9**	**9**	**1304**	**199**	**707**
2440	1¼in.S.-135-55	6034	122	8	451	1082	424	98	423	1762	203	934
2441	1¼in.S.-135-55	6316	144	8	480	1082	231	54	129	1780	147	780
	Average	**6175**	**133**	**8**	**466**	**1082**	**328**	**76**	**276**	**1771**	**175**	**857**

TABLE 26

HEAT BALANCE —PERCENTAGE

Test Number	Laboratory Designation	Per Cent of Heat of Coal as Fired										
		Absorbed by Boiler	To Moisture in Coal	To Moisture in Air	To Hydrogen in Coal	To Escaping Gases	To Incomplete Combustion			To Combustible in Stack Cinders	To Combustible in Ash	To Radiation and Unaccounted for
							CO	H_2	CH_4			
	Code Item ☞	881	882	883	884	885	......	......	......	883	890	899
2400	M. R.-100-33	76.9	0.9	0.4	4.0	11.9	0.4	0.1	0.1	2.2	0.9	2.2
2401	M. R.-100-33	75.0	0.8	0.3	4.1	11.7	0.6	0.0	0.0	2.1	1.7	3.7
2402	M. R.-100-33	75.5	0.9	0.4	4.1	11.2	0.9	0.0	0.2	2.2	0.8	3.8
	Average	**75.8**	**0.9**	**0.4**	**4.1**	**11.6**	**0.6**	**0.0**	**0.1**	**2.2**	**1.1**	**3.2**
2405	M. R.-135-55	64.1	0.9	0.2	4.1	10.5	1.9	0.3	0.7	9.1	2.1	6.1
2406	M. R.-135-55	66.9	0.9	0.2	4.3	10.8	3.5	0.1	0.2	7.9	2.2	3.0
2429	M. R.-135-55	65.1	1.0	0.1	4.2	11.4	1.0	0.1	0.3	8.2	2.6	6.0
	Average	**65.4**	**0.9**	**0.2**	**4.2**	**10.9**	**2.1**	**0.2**	**0.4**	**8.4**	**2.3**	**5.0**
2408	Nut-100-33	73.1	0.9	0.2	4.2	12.5	1.7	0.3	0.4	1.2	0.9	4.6
2409	Nut-100-33	75.0	1.0	0.2	4.2	12.0	1.8	0.2	0.6	1.4	1.3	2.3
2410	Nut-100-33	73.1	0.9	0.2	4.2	11.5	1.4	0.2	0.4	1.5	1.3	5.3
2426	Nut-100-33	73.3	0.8	0.1	4.2	11.3	1.6	0.3	0.4	2.1	0.8	5.1
	Average	**73.6**	**0.9**	**0.2**	**4.2**	**11.8**	**1.6**	**0.3**	**0.5**	**1.6**	**1.1**	**4.3**
2412	Nut-135-55	67.7	1.0	0.1	4.0	10.1	4.8	0.6	1.9	5.3	1.6	2.9
2413	Nut-135-55	67.7	1.0	0.1	3.9	11.1	2.0	0.9	2.0	5.2	0.9	5.2
2414	Nut-135-55	67.2	1.0	0.2	4.2	11.6	2.2	0.5	0.4	5.0	1.8	5.9
	Average	**67.5**	**1.0**	**0.1**	**4.0**	**10.9**	**3.0**	**0.7**	**1.4**	**5.2**	**1.4**	**4.7**
2415	Egg-100-33	74.7	0.9	0.3	4.2	12.6	0.9	0.1	0.4	1.5	1.5	2.9
2416	Egg-100-33	75.1	1.0	0.3	4.3	13.1	0.6	0.0	0.0	1.5	1.6	2.5
2423	Egg-100-33	75.3	1.0	0.2	4.3	11.9	0.7	0.0	0.1	1.4	1.5	3.6
	Average	**75.0**	**1.0**	**0.3**	**4.3**	**12.5**	**0.7**	**0.0**	**0.2**	**1.5**	**1.5**	**3.0**
2420	Egg-135-55	67.5	1.0	0.2	4.3	10.4	1.5	0.1	0.3	6.5	2.9	5.3
2422	Egg-135-55	65.9	1.0	0.2	4.3	11.1	1.3	0.4	0.4	6.2	1.8	7.4
2424	Egg-135-55	66.6	1.0	0.2	4.4	11.0	1.4	0.2	0.4	6.7	2.7	5.4
	Average	**66.7**	**1.0**	**0.2**	**4.3**	**10.8**	**1.4**	**0.2**	**0.4**	**6.5**	**2.5**	**6.0**
2417	Lump-100-33	75.7	1.0	0.2	4.1	12.8	0.5	0.0	0.1	1.4	2.2	2.0
2418	Lump-100-33	73.6	1.1	0.2	4.1	12.5	0.4	0.0	0.0	1.2	2.3	4.6
2419	Lump-100-33	74.2	1.1	0.2	4.2	12.2	0.7	0.1	0.1	1.4	2.6	3.2
	Average	**74.5**	**1.1**	**0.2**	**4.1**	**12.5**	**0.5**	**0.0**	**0.1**	**1.3**	**2.4**	**3.3**
2425	Lump-135-55	61.5	1.0	0.1	4.1	10.9	2.2	0.5	1.1	6.6	0.8	11.2
2427	Lump-135-55	58.9	1.0	0.1	4.2	9.5	4.5	0.1	0.2	7.1	4.7	9.7
2428	Lump-135-55	62.1	1.0	0.1	4.2	10.8	2.6	0.4	0.6	6.2	2.6	9.4
2442	Lump-135-55	63.2	1.0	0.1	4.3	11.0	1.1	0.1	0.3	6.9	3.3	6.7
	Average	**61.9**	**1.0**	**0.1**	**4.2**	**10.6**	**2.6**	**0.3**	**0.6**	**6.7**	**2.9**	**9.3**
2430	2 in. S.-100-33	70.6	1.1	0.1	4.3	10.8	1.2	0.2	0.5	6.7	1.2	3.3
2434	2 in. S.-100-33	70.2	1.1	0.1	4.4	11.3	0.6	0.0	0.2	7.3	1.6	3.2
2435	2 in. S.-100-33	70.8	1.0	0.1	4.4	11.1	1.1	0.0	0.0	7.4	1.0	3.1
	Average	**70.5**	**1.1**	**0.1**	**4.4**	**11.1**	**1.0**	**0.1**	**0.2**	**7.1**	**1.3**	**3.2**
2436	2 in. S.-135-55	56.3	1.1	0.1	4.5	11.7	1.7	0.3	0.8	12.1	1.9	9.5
2437	2 in. S.-135-55	57.4	1.1	0.1	4.5	10.0	1.9	0.4	0.5	14.3	1.7	8.1
	Average	**56.9**	**1.1**	**0.1**	**4.5**	**10.9**	**1.8**	**0.4**	**0.7**	**13.2**	**1.8**	**8.8**
2431	1¼ in. S.-100-33	62.8	0.9	0.2	4.3	9.8	1.2	0.0	0.0	11.9	1.7	7.2
2432	1¼ in. S.-100-33	66.1	1.0	0.2	4.3	10.2	0.8	0.1	0.1	11.6	1.7	3.9
2433	1¼ in. S.-100-33	64.2	0.9	0.2	4.3	10.7	0.9	0.2	0.2	10.0	1.8	6.6
	Average	**64.4**	**0.9**	**0.2**	**4.3**	**10.2**	**1.0**	**0.1**	**0.1**	**11.2**	**1.7**	**5.9**
2440	1¼ in. S.-135-55	52.3	1.1	0.1	3.9	9.4	3.7	0.9	3.7	15.3	1.8	7.8
2441	1¼ in. S.-135-55	56.6	1.3	0.1	4.3	9.7	2.1	0.5	1.2	16.0	1.3	6.9
	Average	**54.5**	**1.2**	**0.1**	**4.1**	**9.6**	**2.9**	**0.7**	**2.5**	**15.7**	**1.6**	**7.4**

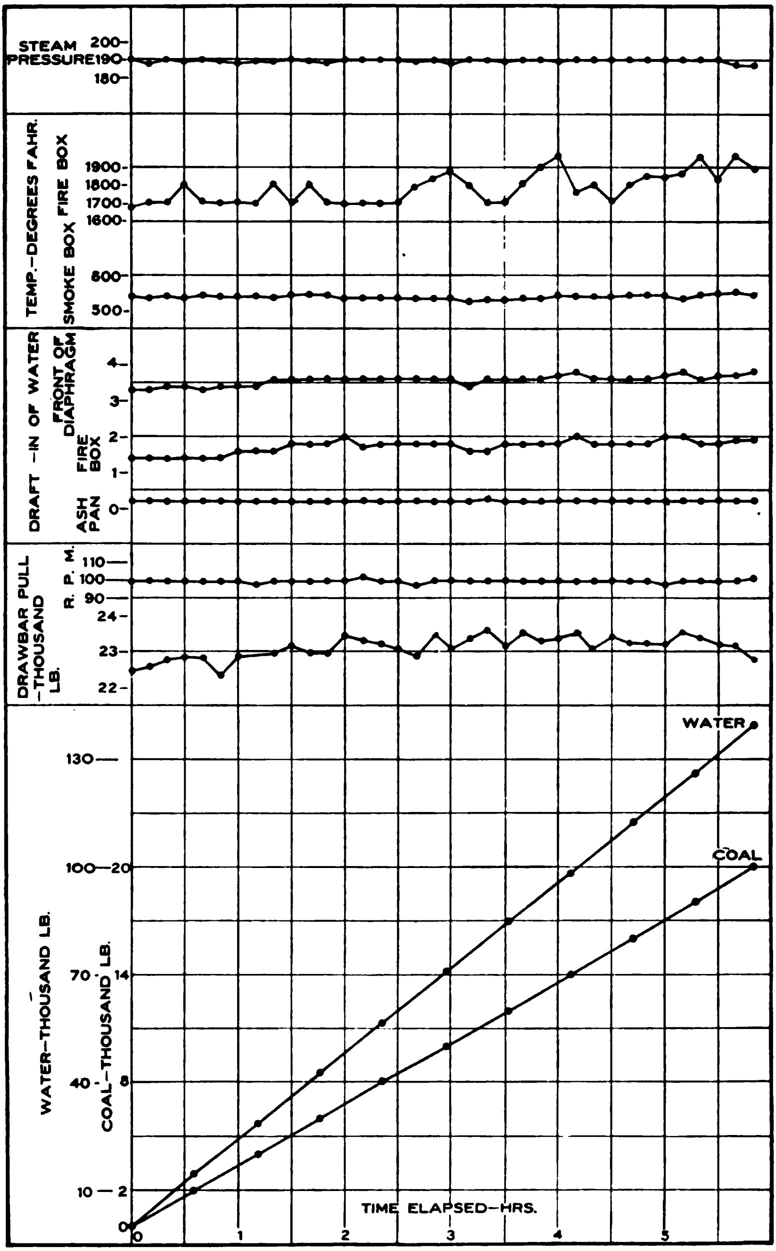


FIG. 29. GRAPHICAL LOG FOR MEDIUM RATE TEST NO. 2416

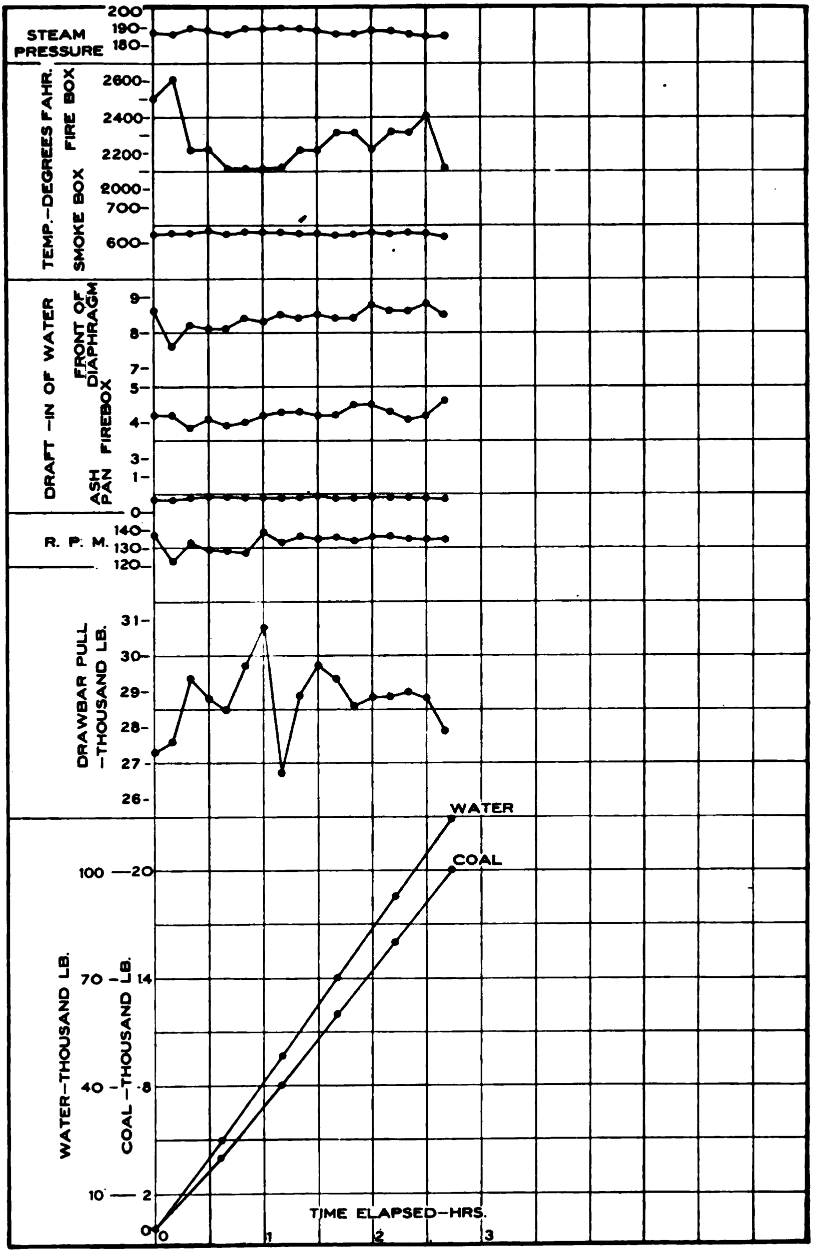


Fig. 30. Graphical Log for High Rate Test No. 2405

APPENDIX IV

CYLINDER PERFORMANCE

The main purpose of the tests entailed the definition of the boiler performance at only two rates of evaporation. This necessitated tests under only two conditions of speed and cut-off, which presented but little opportunity to gather data concerning the performance of the cylinders. Indicator cards were taken during all tests, under these conditions; but it proved impracticable to hold the locomotive for the determination of cylinder performance at other speeds and cut-offs, as had been originally intended. The available facts concerning cylinder performance are presented in detail for each of the tests in Tables 17, 21 and 22 of Appendix III, and their average values at both combinations of speed and cut-off are stated in the following sections. Representative indicator cards are reproduced in Fig. 31, and the data relating to them appear in Table 27.

32. *Medium Rate Tests.*—All medium rate tests were run with the reverse-lever in the second notch from the center of the quadrant, giving a cut-off of about 33 per cent. The speed was maintained as nearly as possible at 100 revolutions per minute, which is equivalent to 19.0 miles per hour on the road.

The average indicated horse power was 1305. It varied from 1224 to 1365.

The average drawbar pull was 22640 pounds. The average indicated horse power consumed in machine friction was 160.

The steam consumed per indicated horse power per hour varied from a minimum of 17.73 pounds to a maximum of 18.74 pounds; and the average for all medium rate tests was 18.18 pounds.

33. *High Rate Tests.*—All high rate tests were run with the reverse-lever in the sixth notch from the center, giving a cut-off of about 55 per cent. The speed was maintained as nearly as possible at 135 revolutions per minute, which is equivalent to 25.7 miles per hour.

The average indicated horse power was 2196. It varied from 2126 to 2243.

The average drawbar pull was 28826 pounds. The average indicated horse power consumed in machine friction was 223.

The steam consumed per indicated horse power per hour varied from a minimum of 19.00 pounds to a maximum of 20.24 pounds; the average for all high rate tests was 19.58 pounds.

34. *Variations in Power.*—Despite the fact that both the reverse-lever position and the speed were maintained constant during all medium rate and during all high rate tests, there was during the progress of the work considerable variation in indicated horse power in both groups. In general the power increased as time went on. An almost identical variation occurs in the areas of the indicator cards. Neither the variations in water rate nor in superheat offer an adequate explanation for these facts. It is assumed that they are due chiefly to changes in steam distribution brought about by wear in the valve gear.

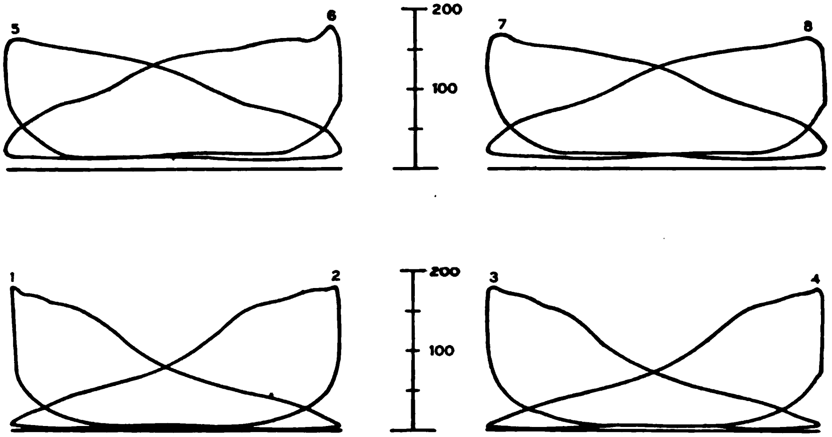


FIG. 31. REPRESENTATIVE INDICATOR DIAGRAMS FOR BOTH THE MEDIUM AND THE HIGH RATE TESTS

TABLE 27

INFORMATION CONCERNING THE INDICATOR DIAGRAMS SHOWN IN FIG. 37

TEST NUMBER	Laboratory Designation	Diagram No.	Right or Left Side	Head or Crank End	Average Cut-off for Test Per Cent	Average Speed for Test	
						Miles per Hour	Revolutions per Minute
2416	Egg-100-33	1	R	H	31.0	18.94	99.5
2416	Egg-100-33	2	R	C	33.8	18.94	99.5
2416	Egg-100-33	3	L	H	36.0	18.94	99.5
2416	Egg-100-33	4	L	C	32.3	18.94	99.5
2405	M. R.-135-55	5	R	H	49.9	25.47	133.8
2405	M. R.-135-55	6	R	C	58.8	25.47	133.8
2405	M. R.-135-55	7	L	H	55.0	25.47	133.8
2405	M. R.-135-55	8	L	C	53.4	25.47	133.8

APPENDIX V

COMPARISON OF LONG AND SHORT TESTS

In connection with this series of tests, the question arose as to the desirability or necessity of running tests of such a length that the amount of coal burned would be about the same as a freight locomotive would burn in making a trip over an ordinary railroad division. This included questions such as that concerning the relative reliability of short tests burning from 4 to 6 tons of coal, and of longer tests burning from 10 to 12 tons; and that of the relative performance during the first, middle and last parts of a test burning 10 or 12 tons of coal.

In order to determine to some extent the difference in boiler performance which might occur during short and long tests and during different parts of a test, the observations were so taken that the exact amount of water evaporated by the boiler could be determined for intervals corresponding to the firing of each 2000 pounds of coal during the medium rate tests, and corresponding to the firing of each 4000 pounds of coal during the high rate tests.

For the purpose of making comparisons, six tests—three at the medium rate and three at the high rate—have been selected and divided into shorter tests. From 8½ to 10½ tons of coal were burned during each of these six tests. The test data for each entire test were divided into three parts, resulting in data for eighteen comparatively short tests together with the data for the six original tests. With the data and results of the 24 tests thus obtained it is possible to make the following comparisons: First, between long and short tests; second, between the first, middle and last portions of a given test; and third, between the different portions and the entire test.

Table 28 presents the significant data and results for the 24 tests. Six groups of four tests each appear in the table. In each group sections *a*, *b* and *c* present, respectively the first, middle, and last portion of the entire test, developed as separate and distinct units. Immediately following the data for these three portions appear the corresponding results for the entire test.

The length of the different tests is shown in Columns 3, 4, 5 and 6 of Table 28. The entire medium rate tests were approximately 6 hours long and the entire high rate tests 3 hours long. The divided tests varied in length from about one-half hour to two and one-half hours. From 224 to 271 pounds of coal were burned per square foot of grate during each entire test while for the divided tests the corresponding amounts vary from 52 to 115 pounds of coal per square foot of grate. The equivalent evaporation per square foot of heating surface varied for the entire tests from 33 to 40 pounds and for the divided tests from 6 to 16 pounds. At the University of Illinois locomotive laboratory about 150 pounds of coal burned per square foot of grate, or from 15 to 20 pounds of equivalent evaporation per square foot of heating surface, have been considered as sufficient to avoid any serious errors in test results arising from inaccuracies in coal and water measurements.

TABLE 28

TEST CONDITIONS AND PRINCIPAL RESULTS FOR SIX TESTS, WHICH HAVE BEEN DIVIDED INTO THREE TESTS EACH

1	2	3	4	5	6	7	8	9	10	11	12
		Length of Test					Pressure lb. per sq. in.		Temperature Degrees F.		
TEST NUMBER RATE AND SIZE	Section of Test or Entire Test	Minutes	Hours	Dry Coal Burned per Sq. Ft. of Grate During the Test lb.	Equivalent Evaporation per Sq. Ft. of H.S. During the Test lb.	Speed in Miles per Hour	Boiler Gauge	Branch-pipe Gauge	Front-end	Branch-pipe	Fire-box
Code Item ☞		....	345	....	....	353	380	383	367	370	374
2401	a	114	1.90	79.2	12.0	18.9	190.0	172	539	566	1774
Medium	b	151	2.52	105.6	16.4	19.0	190.0	172	533	564	1835
Mine	c	112	1.87	79.2	11.3	18.8	190.3	172	532	566	1907
Run	**Entire**	**377**	**6.28**	**264.1**	**39.7**	**18.9**	**190.0**	**172**	**535**	**566**	**1835**
2402	a	107	1.78	79.0	11.7	19.1	190.4	176	537	565	1772
Medium	b	115	1.92	79.0	12.4	18.9	190.1	172	542	563	1847
Mine	c	90	1.50	65.9	9.6	18.9	190.1	173	537	565	1825
Run	**Entire**	**312**	**5.20**	**223.9**	**33.7**	**19.0**	**190.2**	**174**	**539**	**564**	**1812**
2416	a	106	1.77	78.1	12.0	19.0	189.3	180	541	574	1852
Medium	b	141	2.35	104.1	15.7	18.8	189.5	180	536	572	1759
3″ x 6″	c	103	1.72	76.9	11.7	19.0	189.5	180	544	577	1794
Egg	**Entire**	**350**	**5.83**	**259.1**	**39.4**	**18.9**	**189.5**	**180**	**540**	**574**	**1801**
2405	a	69	1.15	105.2	14.0	25.1	189.1	168	628	628	2284
High	b	31	0.52	52.6	6.4	25.2	188.7	169	627	628	2238
Mine	c	63	1.05	105.2	12.9	26.0	185.9	168	625	628	2279
Run	**Entire**	**163**	**2.72**	**262.9**	**33.3**	**25.5**	**187.8**	**188**	**627**	**628**	**2271**
2406	a	72	1.20	105.3	14.7	25.6	187.8	167	631	629	2257
High	b	34	0.57	52.6	7.0	25.7	188.5	169	632	630	2320
Mine	c	55	0.92	87.2	11.0	25.7	187.1	168	630	634	2467
Run	**Entire**	**161**	**2.68**	**245.1**	**32.7**	**25.6**	**187.8**	**167**	**631**	**631**	**2334**
2413	a	72	1.20	104.3	14.8	25.7	187.3	168	616	628	2227
High	b	34	0.57	52.2	7.3	25.7	186.5	168	615	632	2271
2″ x 3″	c	74	1.23	114.9	15.2	25.7	187.3	168	602	636	2305
Nut	**Entire**	**180**	**3.00**	**271.3**	**37.3**	**25.7**	**187.1**	**168**	**611**	**632**	**2267**

TABLE 28 (Continued)

TEST CONDITIONS AND PRINCIPAL RESULTS FOR SIX TESTS, WHICH HAVE BEEN DIVIDED INTO THREE TESTS EACH

1	2	13	14	15	16	17	18	19	20	21	22
Test Number Rate and Size	Section of Test or Entire Test	Draft in. of Water			Draw-bar Pull lb.	Degrees of Superheat	Coal as Fired lb.			Dry Coal lb.	
		Front-end Front of Diaphragm	Fire-box	Ash-pan			Total	Per Hour	Per Hour per Sq. ft. of Grate Surface	Per Hour	Per Hour per Sq. ft. of Grate Surface
Code Item ☞		**394**	**396**	**397**	**487**	**409**	**418**	....	....	**626**	**627**
2401 Medium Mine Run	a b c **Entire**	2.7 2.9 2.8 **2.8**	1.3 1.6 1.6 **1.5**	0.2 0.2 0.2 **0.2**	21463 21833 21864 **21727**	190 188 190 **190**	6000 8000 6000 **20000**	3158 3178 3214 **3183**	45.2 45.5 46.1 **45.6**	2911 2929 2962 **2934**	41.7 42.0 42.4 **42.0**
2402 Medium Mine Run	a b c **Entire**	2.9 3.0 3.0 **3.0**	1.4 1.7 1.7 **1.6**	0.2 0.2 0.2 **0.2**	21873 21885 21684 **21822**	187 187 189 **187**	6000 6000 5000 **17000**	3365 3130 3333 **3269**	48.2 44.8 47.8 **46.8**	3094 2877 3065 **3005**	44.3 41.2 43.9 **43.1**
2416 Medium 3″ x 6″ Egg	a b c **Entire**	3.4 3.6 3.7 **3.6**	1.5 1.8 1.9 **1.7**	0.2 0.2 0.2 **0.2**	22782 23262 23231 **23115**	195 193 198 **195**	6000 8000 5915 **19915**	3396 3404 3445 **3414**	48.7 48.8 49.4 **48.9**	3084 3092 3129 **3101**	44.2 44.3 44.8 **44.4**
2405 High Mine Run	a b c **Entire**	8.2 8.5 8.6 **8.4**	4.1 4.3 4.4 **4.2**	0.4 0.4 0.4 **0.4**	28879 28708 28687 **28771**	254 253 254 **254**	8000 4000 8000 **20000**	6957 7737 7619 **7361**	99.7 110.9 109.2 **105.5**	6382 7099 6990 **6753**	91.4 101.7 100.2 **96.8**
2406 High Mine Run	a b c **Entire**	8.6 8.8 8.7 **8.6**	4.2 4.5 4.1 **4.3**	0.4 0.4 0.4 **0.4**	28600 29183 28642 **28718**	255 255 260 **257**	8000 4000 6630 **18630**	6667 7055 7230 **6944**	95.5 101.1 103.6 **99.5**	6122 6480 6640 **6377**	87.7 92.8 95.1 **91.4**
2413 High 2″ x 3″ Nut	a b c **Entire**	8.9 9.2 9.6 **9.2**	4.0 4.4 4.7 **4.4**	0.5 0.5 0.5 **0.5**	28791 29407 29294 **29100**	253 257 261 **258**	8000 4000 8811 **20811**	6667 7055 7146 **6937**	95.5 101.1 102.4 **99.4**	6067 6420 6503 **6313**	86.9 92.0 93.2 **90.4**

Examination of the data shows that test conditions with regard to speed, pressures, temperatures, drafts, drawbar pull, and quality of steam were very uniform as between the first, middle, and final sections of each test, with the single exception of fire-box temperature. In general the temperature in the fire-box increased somewhat as the test proceeded.

Columns 22 and 29 show, respectively, the rate of combustion expressed in dry coal per square foot of grate per hour, and the rate of evaporation expressed in equivalent evaporation per square foot of heating surface per hour. During the medium rate tests these rates were quite uniform through the 3 sections of each entire test. During the high rate tests the rate of combustion increased as each test proceeded, the greater part of this increase occurring during the first

TABLE 28 (Concluded)

TEST CONDITIONS AND PRINCIPAL RESULTS FOR SIX TESTS, WHICH HAVE BEEN DIVIDED INTO THREE TESTS EACH

1	2	23	24	25	26	27	28	29	30	31	32
TEST NUMBER RATE AND SIZE	Section of Test or Entire Test	Quality of Steam in Dome	Superheated Steam lb.				Equivalent Evaporation lb.			B.t.u. Absorbed by Boiler per lb. of Coal as Fired	Boiler Efficiency Per Cent
			Per Hour	Per Sq. ft. of Heating Surface per Hour	Per Pound of Coal as Fired	Per Pound of Dry Coal	Per Hour	Per Sq. ft. of Heating Surface per Hour	Per Pound of Dry Coal		
Code Item ☞		**407**	....	....	....	....	**645**	**648**	**656**	....	**666**
2401	a	0.9773	22405	4.81	7.09	7.70	29507	6.33	10.14	9076	75.68
Medium	b	0.9798	22973	4.93	7.23	7.85	30278	6.50	10.34	9260	77.22
Mine	c	0.9802	21390	4.59	6.66	7.23	28213	6.05	9.53	8532	71.14
Run	**Entire**	**0.9791**	**22328**	**4.79**	**7.01**	**7.61**	**29451**	**6.32**	**10.04**	**8968**	**74.95**
2402	a	0.9789	23217	4.98	6.90	7.51	30440	6.53	9.84	8794	73.99
Medium	b	0.9803	22961	4.92	7.33	7.98	30217	6.48	10.50	9377	78.90
Mine	c	0.9807	22710	4.87	6.82	7.41	29886	6.41	9.75	8716	73.34
Run	**Entire**	**0.9801**	**22970**	**4.93**	**7.02**	**7.64**	**30183**	**6.43**	**10.04**	**8969**	**75.46**
2416	a	0.9888	24037	5.16	7.08	7.80	31729	6.81	10.29	9076	76.15
Medium	b	0.9883	23557	5.05	6.92	7.62	31143	6.68	10.07	8891	74.60
3″ x 6″	c	0.9886	23952	5.14	6.95	7.65	31736	6.81	10.14	8949	75.09
Egg	**Entire**	**0.9885**	**23788**	**5.11**	**6.97**	**7.67**	**31448**	**6.75**	**10.14**	**8949**	**75.09**
2405	a	0.9612	41962	9.01	6.03	6.58	56649	12.16	8.88	7910	67.30
High	b	0.9622	42155	9.09	5.48	5.96	57306	12.30	8.07	7200	61.27
Mine	c	0.9660	42176	9.05	5.54	6.03	57107	12.25	8.17	7288	62.01
Run	**Entire**	**0.9628**	**42176**	**9.05**	**5.73**	**6.24**	**57022**	**12.24**	**8.44**	**7531**	**64.06**
2406	a	0.9579	42309	9.08	6.35	6.91	57201	12.27	9.34	8337	70.20
High	b	0.9607	42321	9.08	6.00	6.53	57345	12.31	8.85	7900	66.52
Mine	c	0.9549	41200	8.84	5.70	6.20	55909	12.00	8.42	7498	63.14
Run	**Entire**	**0.9574**	**41946**	**9.00**	**6.04**	**6.58**	**56795**	**12.19**	**8.91**	**7949**	**66.93**
2413	a	0.9449	42374	9.09	6.36	6.98	57332	12.30	9.45	8357	69.70
High	b	0.9478	44272	9.50	6.28	6.90	60077	12.89	9.36	8279	69.05
2″ x 3″	c	0.9489	42381	9.09	5.93	6.52	57553	12.35	8.85	7822	65.24
Nut	**Entire**	**0.9470**	**42720**	**9.17**	**6.16**	**6.77**	**57929**	**12.43**	**9.18**	**8114**	**67.67**

section. During the high rate tests the rate of evaporation increased with the increasing rate of combustion during the first part of the test, but decreased during the latter part. This indication of somewhat poorer performance during the latter part of a long test is shown more exactly by the values relating to efficiency.

Columns 30, 31 and 32 show the efficiency of the locomotive boiler as a heat transferring device. For the medium rate tests the results indicate that the efficiency during the first and middle sections of the tests was higher than during the last section. For two out of three of the medium rate tests the middle section showed a materially higher efficiency than either the first or last sections. For the high rate tests efficiency decreased in general from the first to the last section of each test, showing the best performance during the first part of the test and poorer performance as the test proceeded.

The differences in performance which have been pointed out as existing between different parts of a long test are in general small. Where test conditions are not unusual and are under control, as is the case in a testing laboratory, and where it is possible to maintain uniformly good fire-box conditions, short tests should give almost as reliable and almost the same results regarding evaporative performance and efficiency as much longer tests.

Boiler efficiency in general decreases as a test proceeds, so that, in so far as differences in efficiency exist, the average result for a long test would be lower than for a short test corresponding to the first part of the long test, and higher than for a short test corresponding to the last part of the long test. Boiler efficiency is more apt to be uniform throughout long tests at medium rates of combustion than at high rates of combustion.

The coal used during these tests gave little trouble in the firebox, a very small amount of clinkers being formed and the ash being readily removed. With coal which clinkers badly or which produces excessive honeycombing, the variations in performance between different parts of a long test might be much greater than those here shown.

LIST OF
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Bulletin No. 1. Tests of Reinforced Concrete Beams, by Arthur N. Talbot, 1904. *None available.*

Circular No. 1. High-Speed Tool Steels, by L. P. Breckenridge. 1905. *None available.*

Bulletin No. 2. Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks. 1905. *None available.*

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